RESEARCH REPORT

EN3-8967 : RR 90-1

REDUCTION OF FREQUENCY OF SULPHIDE ORE DUST EXPLOSIONS BY IMPROVED BLASTING PRACTICES

(Project No. 09SQ.23440-8-9227)

by

NORANDA TECHNOLOGY CENTRE
J.P. Saindon
Yvan Dionne

JKMRC
Gideon Chitombo
Pat Walker
Mark Mckew

Work on the project was conducted under the auspices of the Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources, Canada

DSS FILE NUMBER: **014SQ.23440-8-9227**
DSS CONTRACT SERIAL NUMBER: **23440-8-9227/01-SQ**
CANMET SCIENTIFIC AUTHORITY: **Dr. K. Mintz**

March 1990

Environmental and Mining Laboratory
Centre de Technologie Noranda
240, boulevard Hymus
Pointe-Claire, Québec
H9R 1G5

97 pp

MICROMEDIA

DISTRIBUTION

CANMET

K. Mintz
D. Dainty
(20 copies)

New Brunswick Government
G. Greer
R. Blanchard

Noranda Minerals

J. Gordon
R. Lauay
J. White
K. Byberg
B. Jamieson
J. Keyes
F. Grebenc
H. Derbuch

Brunswick Mining & Smelting Mining Division

J. Schnarr
F. Hermann
B. Kristoff
G. Crowther
R. Jessulat
JKMRC
G. Chitombo (3)

Noranda Technology Centre

F.Lederman/J.Nantel/Library
K. Wheeland/Dept. file
J.P. Saindon
D. Sprott
A. Dasys
Library

ACKNOWLEDGEMENTS

This work was funded by the New-Brunswick Mineral Development Agreement fund (MDA) and Noranda Minerals Inc. The support of CANMET, in particular Mr. D. Dainty and Dr. K. Mintz of The Explosive Atmosphere Laboratories, in defining and initiating this work was essential.

We would like to thank the mining crew who worked with us during this project (Ron Jessulat, Frank Godin, Cletus Morrisson and Donald Plourde). Their dedication and experience in the field was also essential to the success of the project.

Finally, we received excellent support from Brunswick's engineering department, in particular Glenn Crowther and Jean-Pierre Basque.

KEY WORDS

Blasting
Development Blasting
Dust Explosion
Dust Generation
Flame Propagation
Explosive Flame
Sulphide Dust Explosion
Secondary Dust Explosion
Sulphide Dust
Vibration Monitoring
Video Monitoring
Tunnelling

TABLE OF CONTENTS

	Page
SUMMARY	1
INTRODUCTION	5
1. PHASE 1 - DEVELOPMENT BLASTING REVIEW AND PROJECT PLANNING	
1.1 DEVELOPMENT BLASTING REVIEW	
1.1.1 Bibliographic Review	6
1.1.2 Computer Modeling	7
1.1.3 Development Blasting	9
1.2 PROJECT PLANNING	
1.2.1 The Mine and the Test Area	20
1.2.2 Instrumentation	20
1.2.3 Schedule	24
2. PHASE 2 - FIELD WORK	
2.1 INITIAL PREPARATION	
2.1.1 Project Timing	25
2.1.2 Objectives	25
2.1.3 Test Area and Crew	26
2.2 BLAST AUDITING (STAGE 1)	
2.2.1 General	27
2.2.2 Monitored Blast Descriptions and Locations	27
2.2.3 Design Parameters	34
2.2.4 Blast Performance	36
2.2.5 Explosive Distribution	46
2.2.6 Summary	46

TABLE OF CONTENTS (CONT'D)

	Page
2.3 TIMING (STAGE 2)	
2.3.1 General	48
2.3.2 Experiments Descriptions and Locations	48
2.3.3 Results	53
2.3.4 Summary	61
2.4 ROUND DESIGN AND PRODUCTION TESTS (STAGE 3)	
2.4.1 General	62
2.4.2 Experiments Description	62
2.4.3 Results	62
2.4.4 Summary	63
3. DISCUSSION OF RESULTS	
3.1 SECONDARY SULPHIDE DUST IGNITIONS	65
3.2 FUTURE DIRECTION	69
REFERENCES	70
APPENDICES	
A - Memo to Project Management on Final Project Status, 3 November 1989	
B - Details of Field Work at Brunswick Mining, 6 August, 9 November 1989	

LIST OF FIGURES

Figure 1	Bench Blast Design
Figure 2	Development Blast Design
Figure 3	Type of Caps Used in the Mining Industry
Figure 4	Effect of Diameter and Distance of Relief and Charged Holes on Blasting Efficiency
Figure 5	Section View of the Three Main Types of Cuts in Development Blasting
Figure 6	Rock Drilling Generations
Figure 7	Instrumentation Set-up in a Heading
Figure 8	Recorder Set-up
Figure 9	Lighting System Set-up
Figure 10	Approximate Location of Blasts: 2 Sub North
Figure 11	Approximate Location of Blasts: 2 Sub North Hangingwall
Figure 12	Approximate Location of Blasts: 1 Sub North
Figure 13	Approximate Location of Blasts: 2 Sub South
Figure 14	Drill Hole Layout: BMS04
Figure 15	Drill Hole Layout: BMS29
Figure 16	Location of Stage 2 Experiments: 2 Sub North
Figure 17	Cut Experiments: 249-8 North Stope Wall
Figure 18	Location of Experiment BMS15
Figure 19	Drill Hole Layout: BMS14
Figure 20	Hole Layout as Drilled: Corkscrew Design
Figure 21	Spray System Set-up

LIST OF TABLES

Table 1	Project Timing
Table 2	Description of Blasts Monitored - Stage 1
Table 3	Description of Blasts Monitored - Stage 3
Table 4	Drilling Details
Table 5	Delay Scatter Analysis
Table 6	Results of Vibration Analysis - Stage 1
Table 7	Results of Vibration Analysis - Stage 3
Table 8	Vibration Results Summarized by Delay Number - Stage 1
Table 9	Vibration Results by Delay Number - Stage 3
Table 10	Development Advance Achieved per Round
Table 11	Description of Blasts Monitored - Stage 2
Table 12	Results of Vibration Analysis
Table 13	Depth of Pull - Trial Cuts

LIST OF PHOTOGRAPHS

Photograph 1 Calculated Explosive Distribution From 3*3D
Photograph 2 Control Cut Design
Photograph 3 Experimental Result of Control Cut Design
Photograph 4 JKMRC Corkscrew Design Trialled at BMS-M
Photograph 5 Experimental Result of JKMRC Corkscrew Design

SUMMARY

The Noranda Technology Centre organized and directed a field study to reduce the frequency of sulphide dust explosions by improved blasting techniques[1]. The Julius Kruttschnitt Mineral Research Center (JKMRC) was sub-contracted by NTC to provide assistance with the blast design and monitoring. The work was funded by The New Brunswick Mineral Development Agreement and Noranda Minerals Inc.

The research was separated into two phases. Phase 1 consisted of undertaking a review of tunnel blasting and preparing the instrumentation for the field work. Phase 2 consisted of the field work carried out at Brunswick No. 12 Mine over a 12 week period between August and November 1989, and was broken into three stages.

Stage 1 monitored current blasting practice in particular and investigated the occurrences of misfires and incomplete detonations, as these were suspected of being one cause of dust ignitions.

Stage 2 was aimed at evaluating short period delays (ETI 25ms delay series), in terms of their ability to reliably provide full advance for development rounds and their potential in avoiding or minimizing sulphide dust ignitions.

Stage 3 provided an opportunity for testing of new cut design as well as a limited evaluation of alternative explosives.

PHASE 1 - LITERATURE REVIEW

The literature review provided limited information on development blasting. Most of the information covered civil engineering work in tunneling which, although relevant to development blasting, was not directly applicable because of the different scale factor.

A recent report on Canadian mines development practices published by Laurentian University[2] demonstrates that techniques are quite consistent throughout the operations and that the round using the Canadian burn cut design and fully loaded with ANFO is the most frequent approach.

Fundamental research on cut design published by Kilhström and Langefors[3] provided valuable information on the efficiency and distinction of burn cuts and parallel cuts. The importance of the geology on the efficiency of these cuts is discussed in Phase 2 of this report.

A brief but complete review of development blasting based on the literature search is provided in the first part of this project.

PHASE 2 - BLASTING TECHNIQUES

Results of Phase 2 showed that different crews monitored throughout the mine were comparable with respect to development blast practices. Most of the blasts and designs monitored were that of a designated Brunswick Mines development crew headed by Ron Jessulat. The crew members were Frank Godin and Donald Plourde. Cletus Morrison assisted during the initial stages of the project.

The blast monitoring work and subsequent experiments were designed around two basic assumptions:

i) Sulphide dust explosions were presumed to be more likely a result of poor blasting practices (and designs), and subsequent poor blast performance, in particular the occurrence of misfires and deflagrating (low order detonating) charges. The above phenomena were expected to produce "lasting" flames which could ignite the dust. This was provided the other conditions were met, i.e. sufficient oxygen concentration and dust concentration.

ii) The timing or speed of extraction of a development round was thought to be a critical factor. The rationale behind the use of short period delays was that insufficient time would elapse for a large reactive dust cloud to form before the last charge in the round (or the cut) had detonated.

Approximately 44 charge and 3 relief holes were drilled per round to provide acceptable advance irrespective of the rock/grain combination. Timing results showed the long period Nonel initiators being used to be of sufficient accuracy with little risk of out of sequence detonations.

The **misfires and poor detonations** recorded during Stage 1 constituted a small proportion 6% of the holes monitored. A total of 497 holes from 13 blasts were recorded. Crews monitored during Phase 2 recorded a higher percentage (13%). The combined total (8%) represents an average of 3 holes per round. Reduction of the number of misfires and deflagrations would require a major project due to their random occurrences across the blasts; total elimination is not considered a realistic goal and of little benefit. Improvements in the rounds being drilled are possible but at most only 2 or 3 holes could be deleted from the pattern.

Short period delays (Stage 2) did not offer any advantages both from the rock breakage and from the sulphide dust explosion elimination perspective.

During these timing trials, it was observed that cuts initiated with short delays produced even finer and therefore more reactive dust. Furthermore, the dust was thrown further into the drives, with the risk of explosions covering a much bigger area.

From a practical or operational point of view, **cut experiments** using short periods delays did not pull the ground to full depth. This included the JKMRC cork screw and some of the proven cut designs used by the Brunswick crew. This appeared to be a function of the Brunswick rock type; the rock seems to require relief and relatively long delay time to successfully pull the cut, and true burn cuts did not seem to work.

In Stage 3, both the JKMRC corkscrew and a double spiral cut were successfully implemented in production using long delays. These cuts have the advantage of reducing drilling time but also require more accuracy than a standard cut. The level of drilling accuracy which can be obtained with pneumatic jumbos is poor and these cuts could therefore not be implemented on a large scale at Brunswick.

Two **alternative explosives**, Trimrite (a water-gel based explosive) and Lomex (an ANFO and polystyrene mixture), were also tested to control the back and reduce the misfires in the cut. The decoupling effect of the Trimrite cartridges (32mm) was successfully used to eliminate the misfires of the helpers while the Lomex was successful at controlling the back in a dike formation. A total of 5 productio blasts were carried out in a high sulphur production zone using Trimrite and no sulphur blasts were encountered. (The water-sprays were not used). This is not a large database and more tests should be conducted in a further project.

SUPPLEMENTARY INVESTIGATIONS: WATER SPRAY AND SLURRIES

In parallel to the work described above on blasting techniques, the efficiency of the water sprays and inert water gel stemming was evaluated for two crews on the 1000 mL no. 2sub.

The results show that both inert water gel stemming and water sprays are not 100% efficient at reducing sulphur dust explosions. In high sulphur level zones (>35%), the water spray was more efficient than the stemming at reducing the damages of the sulphur blast. In a medium risk zone (25-30% sulphur) it almost entirely eliminated the occurrences.

Ron Jessulat's development crew carried out some further tests in November and December which indicated that sulphur blasts could be eliminated in high risk areas by using a combination of the water spray and inert water gel cartridges. This work will continue in 1990 and transfer of the technology has been initiated by Don Gagnon (training coordinator at BMS-M).

KEY CONCLUSIONS AND RECOMMENDATIONS

Analysis of both the video results and vibration traces suggested that the reactive dust cloud was ignited by holes detonating correctly (not misfires or deflagrations) and there were also incidents of subsequent ignitions, possibly by hot gases or hot reactive dust ejected from the hole. The ignition time of the recorded dust explosion was between charge hole detonation. Extinguishing or prevention of the ignition source (rather than minimization of the amount of dust generated on firing) therefore appears to be the more likely means to avoid explosions.

The use of low heat producing explosives, or alternatively a system to remove the heat from the dust ejected, warrant further evaluation. The water sprays as designed by Don Gagnon show potential for lowering the temperature of gas and dust ejected from holes in medium risk sulphur areas. When combined with water gel stemming (only the perimeter holes are required) they appear to be also efficient in high risk areas. This work will be continued by Brunswick Mining, and detailed documentation of the results by various development crews is recommended in 1990.

The second prospect to investigate further is the use of low temperature explosives such as water gel, slurries and permitted explosives. With these products the ejected dust and gases may be at low enough a temperature that dust ignitions are avoided. Any trial of different explosives should be conducted over an extended period to ensure several blasts in a very high risk area are included.

INTRODUCTION

Because of the increasing incidence of sulphide dust explosions in Canadian mines, the study to inhibit such explosions has been the subject of increasing emphasis by both government and industry. Sulphide ore dust explosions occur mainly during blasting operations. A dust cloud is created by a combination of the pressure front from the explosion, raising dust from the surfaces as it travels, or by the explosive charge creating the dust during the blast. This dust cloud can then be ignited by hot gases from the blast.

In the past, most of the work in controlling development sulphide blasts has been focused on methods to reduce or dilute the dust (foggers, lime) and quench the flame (water or water gel cartridges). Recent work carried out by NTC[1] and the USBM[4] has demonstrated that sulphide dust explosions might be more effectively controlled by optimizing development drilling pattern, delay sequence and explosive type.

The Julius Kruttschnitt Mineral Research Center (JKMRC) of Australia has shown, through work carried out in Chilean mines[5,6,7], that most development rounds tend to be overcharged. Overcharging leads to interaction between charges, low efficiency detonation and, as a consequence, longer flame propagation. The interaction between charges can be minimized by changing the drilling pattern and the total number of holes.

The sequence and timing at which the holes are fired are also critical. The burn cut when blasted produces a large percentage of total dust. In a blast which uses long delay caps between holes, the dust from the burn cut is still suspended when the auxiliary and perimeter holes are fired. The flame propagating from these holes can readily ignite the suspended dust. By decreasing the delay to a minimum between holes, dust/flame contact could be minimized.

Some tests by the USBM in coal shale have shown that emulsion blends (ANFO/emulsion mix) and water gel have a lower flame propagation than ANFO[4]. Full scale tests in development oil shale drifts have shown that water-gel will not ignite the dust. Water-gel is used extensively in tunneling work and can either be pumped in horizontal holes using a pump loader and a linker agent or loaded in packaged form with a pneumatic cartridge loader.

It was therefore proposed to carry out a special research effort in development blasting in order to evaluate the potential for reduction of secondary dust explosion risks.

PHASE 1 - DEVELOPMENT BLASTING REVIEW AND PROJECT PLANNING

1.1 DEVELOPMENT BLASTING REVIEW

1.1.1 Bibliographic Review

A review of tunnel blasting was undertaken at the start of the project through a computer database search. In general, published information on development blasting was found to be inadequate, probably because technical reports by mine personnel are circulated internally but not published. However, some good papers and relevant information were located.

The Laurentian University has just published a review of drift blasting practices in Canadian underground mines[2]. This report illustrates the numerous problems which face the mining engineer when designing drift patterns; problems, such as back control, hole deviation, ground condition and water inflows. It also shows that the practices throughout Canada are quite consistent.

There are few books on development blasting[3,8]. Most of the relevant information is found in technical papers[9,10,11,12,13]. These papers are written for the most part by mine operation personnel for their specific rock types and drift dimensions. Since every mine has a different geology, the information could not be applied directly to the project.

Some practical information was found in tunneling papers. However, the tunneling practices usually involve more holes and superior back control than in mining. The best information came from fundamental research on cut design from Bullock[9], Holmberg[10], Kihlström and Langefors[7].

The computer databases contain almost no information on development blasting. The information covers mainly civil engineering work such as tunneling. Although, the papers published are usually recent, they tend to be more qualitative than quantitative.

In development mining, the blast design is carried out by the miners. They usually determine the best pattern to use in a rock type by trial and error. Since they are not taking notes of their attempts and the engineering department does not get involved in these experiments, the process to find the best pattern is not published. This information is also lost when the miners retire. It is only by working with the mining crews over an extended period of time that it is possible to grasp the reasoning behind the techniques used.

1.1.2 Computer Modeling

There are several computer models available to mining companies for blast design and optimization. The models most used in Canada are BLASTPA (ETI), LDBB (NTC), and Sabrex (ICI). These models are used by mining companies to design up hole or down hole production blast and vertical crater retreat (VCR) blasts. The following paragraphs give a short description of each models.

Blastpa[14]

Blastpa was developed by Dr. R.F. Favreau of the Royal Military College of St-Jean, and Explosives Technologies International (ETI) holds the exclusive rights to its use. The program is based on physical and dynamic mechanisms taking place during a blast such as: the compressive shock wave generated by the expansion of gases at the initiation of a blasthole; the tensile shock wave created by the reflection of the compressive wave on a free face; the expansion of gases which opens secondary cracks and the fragmentation of the rock mass once the secondary cracks reach the free face. Other aspects such as heat transfer, explosive distribution, blasting parameters (hole diameter, stemming, undercut, etc.) types of explosives, rock to explosive coupling the elastic properties of the rock and the initiation sequence are also treated from a mathematical standpoint.

Various subroutines are available within the model. The principal options are as follows:

a) Production blasting by uphole or downhole (including fan drilling).

b) Final contour study of the blast.

c) Progressive study of the free face position and contour throughout the blast.

d) Modified VCR blasting for holes blasted simultaneously (as used at INCO Mines).

e) Interaction of charges study.

f) Overburden casting using explosives (developed for coal mines).

To use the subroutines, the model must first be calibrated by entering the result of the standard pattern presently used at the operation. This has the effect of fixing various "quality factors" to 100%. For the simulation of new patterns, the quality factors are presented as ratio of the initial data. Therefore Blastpa does not provide absolute results, but rather relative results to the initial calibration input.

To run the model, it is required to enter various mechanical properties such as the rock density, Young's modulus, Poisson coefficient, compressive strength, and dynamic resistance in tension (for contouring studies).

LDBB[15]

This software was developed by CANMET. The model is based on the cratering theory developed by Livingston[16] and not on mathematical representations of the physical and mechanical mechanisms taking place.

The cratering theory requires the calibrations of cratering curves from extensive field work. This work consists of blasting a set of specific blasting patterns and depth of charge and measuring the cratering radius and depth.

The model does not contain all the subroutines of Blastpa and it provides the user with absolute instead of relative results. The mechanical properties of the rock are not used in the model. However, these properties are indirectly taken into consideration when calibrating the crater curves. Each set of curves is exclusive to a rock/explosive combination and it can not only take into account mechanical properties but also the structural properties of the rock.

Sabrex[17]

Sabrex is distributed by ICI Explosives (CIL in Canada) and it is based, like Blastpa, on the mathematical representation of the physical and dynamic mechanism of blasting. It enables the user to alter various elements of the blast design, adjusting hole size, spacing, initiation sequence and other factors. It then predicts the blast performance in terms of:

1. Fragmentation size and distribution
2. Muck pile profile
3. Grade fracture and control
4. Collar breakage and flyrock control
5. Backbreak damage
6. Drilling and explosive costs

The model does not contain all the subroutine of Blastpa but requires for input the same knowledge of rock properties. The accuracy of prediction depends on the accuracy of the properties entered and it does not have the convenience of adjusting quality factors (as for Blastpa) or of incorporating the structural rock properties (as for LDBB).

Both LDBB and Sabrex provide good graphic displays and operate in the DOS environment. Blastpa is not as user friendly and can only be run from a mainframe. Mining companies can gain access via modem to the main frame (based in Bloomington U.S.A.) for approximately $20,000 annually.

Development Blasting Simulation

None of the models which have been described provides simulation for tunnel blasting. It would be possible to use some of the information to predict the backbreak damage and efficiency of the slash holes; however the critical part of the blast, the cut, cannot be optimized with these models.

ETI intends to develop a subroutine for tunnel design under Blastpa (based on a Swedish model), however this model will not be helpful for the cut design. The only model used during the present project will be the 3x3 program developed by the JKMRC. This program allows one to both verify the delay sequence (based on normal probability statistics and standard deviation of delays) and calculate the powder distribution in kg/t. This then allows evaluation of the efficiency of the loaded holes and optimization of the total number of blastholes used.

1.1.3 Development Blasting

The following pages summarizes the techniques of development blasting, based on the database search.

Types of Development Blasts

Blasting can be categorized in two general classes: bench blasting and development blasting. Bench blasting is the most important one: open pit and most of the underground mined material is blasted that way. The blast is loaded in a regular pattern (spacing and burden) and the rock always has a free face where it can expand to. The holes are generally fired row by row, with the row nearest to the free face shot first (Fig. 1).

With development blasting there is no free face for the rock to expand towards, thus, the rock mass has to come forward parallel to the holes drilled (Fig 2). The explosive is not used as efficiently as in benching and a good level of experience is required to design a successful blast. The pattern used has to be drilled precisely and the blasting sequence is critical to the success of the blast. The cut holes are drilled very close together and partially loaded with explosive leaving some room to expand into the unloaded holes. The remainder of the rock is then slashed into the new opening.

The critical part of the blast is this cut. If the cut fails, the rest of the blast will not perform as planned. It is then important to have the best pattern to ensure an optimal rate of advance.

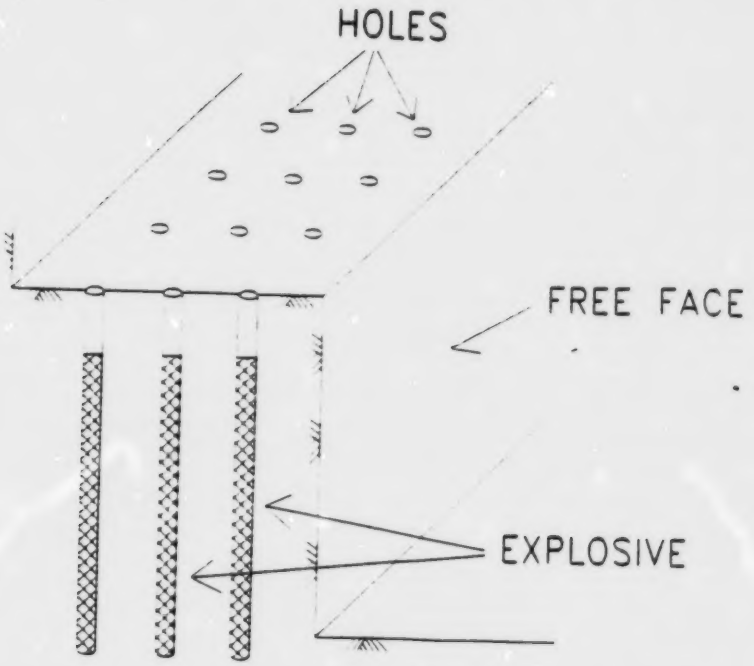


Figure 1 Bench Blast Design

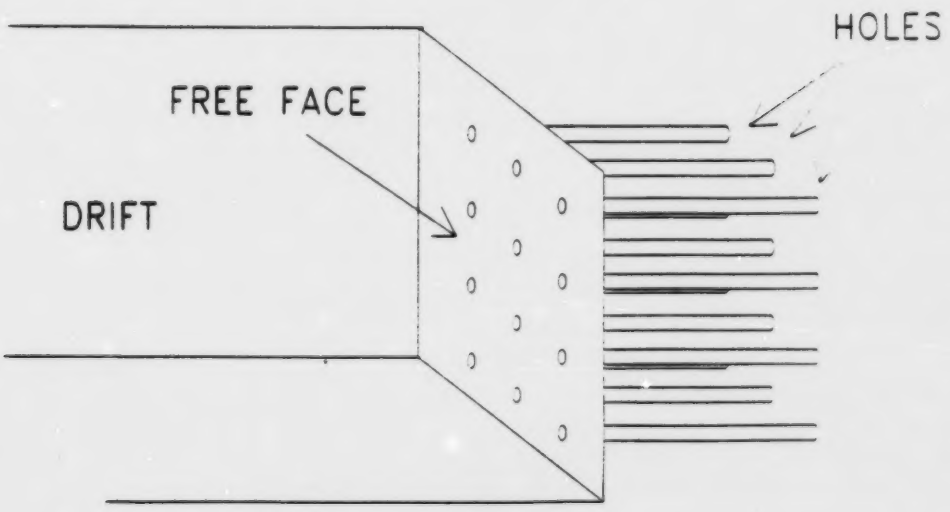


Figure 2 Development Blast Design

The round is generally drilled to the same depth as the smallest dimension of the drift, i.e.: a drift 10 feet wide by 8 feet high will be drilled 8 feet long. The round is composed of different kinds of holes:

Relief: The relief holes are situated in the middle of the cut. They can be of the same size as the charged holes but usually they are reamed to give a wider opening. They are not charged with explosives.

Cut: The cut holes are the first few holes that are fired. Three to six are generally used and their performance is critical for the blast.

Helpers: The helpers are used to widen the cut opening. They are still drilled close together but not as much as in the cut. In some design they are not used.

Stoping: The stoping holes are used to blast the rest of the round. They are situated all around the cut and they blast the bulk of the rock by slashing.

Lifters: The lifter holes are drilled under the round. They consist of a row of holes that will determine the floor elevation.

Back: The back holes are the last row of holes drilled in the top part. They determine the next roof elevation and they are critical to the roof stability. In bad ground, they are usually drilled closer and loaded with less powerful explosive in order to preserve the rock stability.

When a blasted round does not break the rock all the way to the bottom of the holes, it is said that it has not "pulled". The pull or advance is the ratio between the depth drilled and the face advance. This value is generally around 95%, i.e. an 8 foot drilled round will advance 7.6 feet. When full advance is not obtained, some unbroken hole bottoms or "bootlegs" are left in the face and these can be dangerous since undetonated explosive can remain inside.

ANFO

The most popular explosive used in underground mines is ANFO, an ammonium nitrate and fuel oil mixture. Its popularity stems mainly from its safety and low cost. In the round, the holes are loaded with ANFO with a pneumatic loader. The correct practice is to load the hole to within 400mm of the collar and then stem the hole with some inert crushed material. However, more often than not, the explosive is loaded to the collar. Since this explosive column section is not required to blast the rock, it releases additional heat energy in the form of flame and also creates additional fly rocks which can damage vent tubing and hoses in the vicinity.

Types of Caps

Prior to the ANFO loading, a detonating cap is inserted in the bottom of the hole to provide the adequate blasting delay for the hole and act as a primer to initiate the detonation. There are three kinds of detonating caps: tape fuse, electrical and nonel caps. The tape fuse consists of a slow burning tube attached to a #8 type detonating cap. The delay is provided by cutting the tape fuse to the appropriate length. This type of system is not used in any of the Noranda mines anymore (except in a few special cases). An electrical cap uses an electrical impulse from a central switching system to heat an element inside the cap and ignite a pyrotechnic powder. It is the composition and quantity of this powder which determine the time delay of the cap. The powder once ignited will in turn detonate a base PTN charge, powerful enough to ignite the ANFO. Nonel caps (nonelectric) are preferred to electrical caps for their high inherent safety. The cap is similar to the electrical cap, however the electrical wire is replaced by an explosive tube which burns at a very high rate (2000 m/s). This explosive tube can be initiated from underground with a tape fuse or on surface using an electric cap. The system used at Brunswick Mines is an electric detonator which initiates a detonating cord (B-Line) which is attached to all the nonel detonating tubes. Two sets of delays are available for the nonel caps: long and short delays. Long delays are approximately 400 milliseconds apart and short delays 25-50 ms. The accuracy of the delays varies between 5-10%. The three types of caps are illustrated in Figure 3.

Blast Mechanism

The mechanisms to break the rock using explosive can be complex and are not completely understood. When an explosive detonates, it is transformed into a gas within few thousandths of a second. The speed of the reaction can be as high as 6000 m/s. A shock wave is created by the expanding gases and generates intense stress capable of shattering the rock around the opening. Outside of this shattering, radial cracks are created by the tensile stress induced by the explosion. The shock wave travels very fast in the rock. When the explosive gases expand, they enlarge the cracks and shatter the rock. The gases are also responsible for the rock movements because of their intense pressure.

The rock behavior will depend on whether or not a free face is available near the hole. If there is no free face, the explosive gases will rifle through the hole opening. Only a small area around the hole will be highly crushed and displaced. In the case of an unconfined hole, the explosive column detonates efficiently and breaks the rock to the free face. The rock is ejected out of the hole. A relation between the charged hole diameter and the relief hole diameter has been suggested by Langefors and Kihlström[3]. To have a clean blast between the two holes, the distance must not be greater than 1.5 times the diameter of the relief hole. If this distance is increased there will only be breakage of the rock with no ejection (Fig. 4).

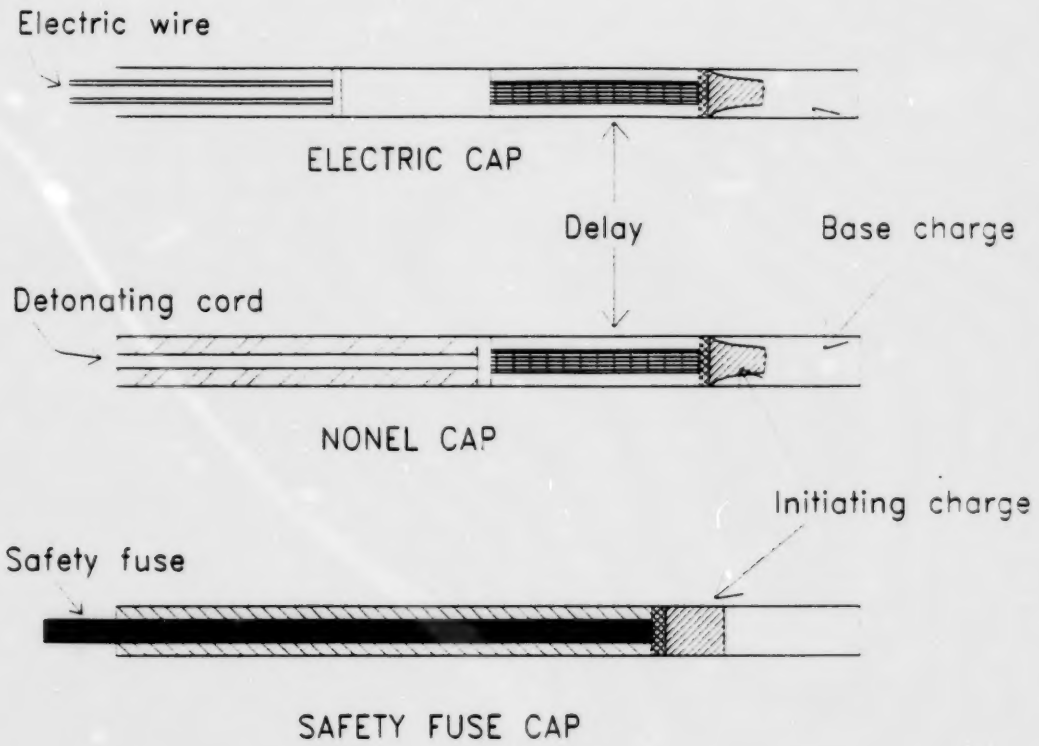


Figure 3 Type of Caps Used in the Mining Industry

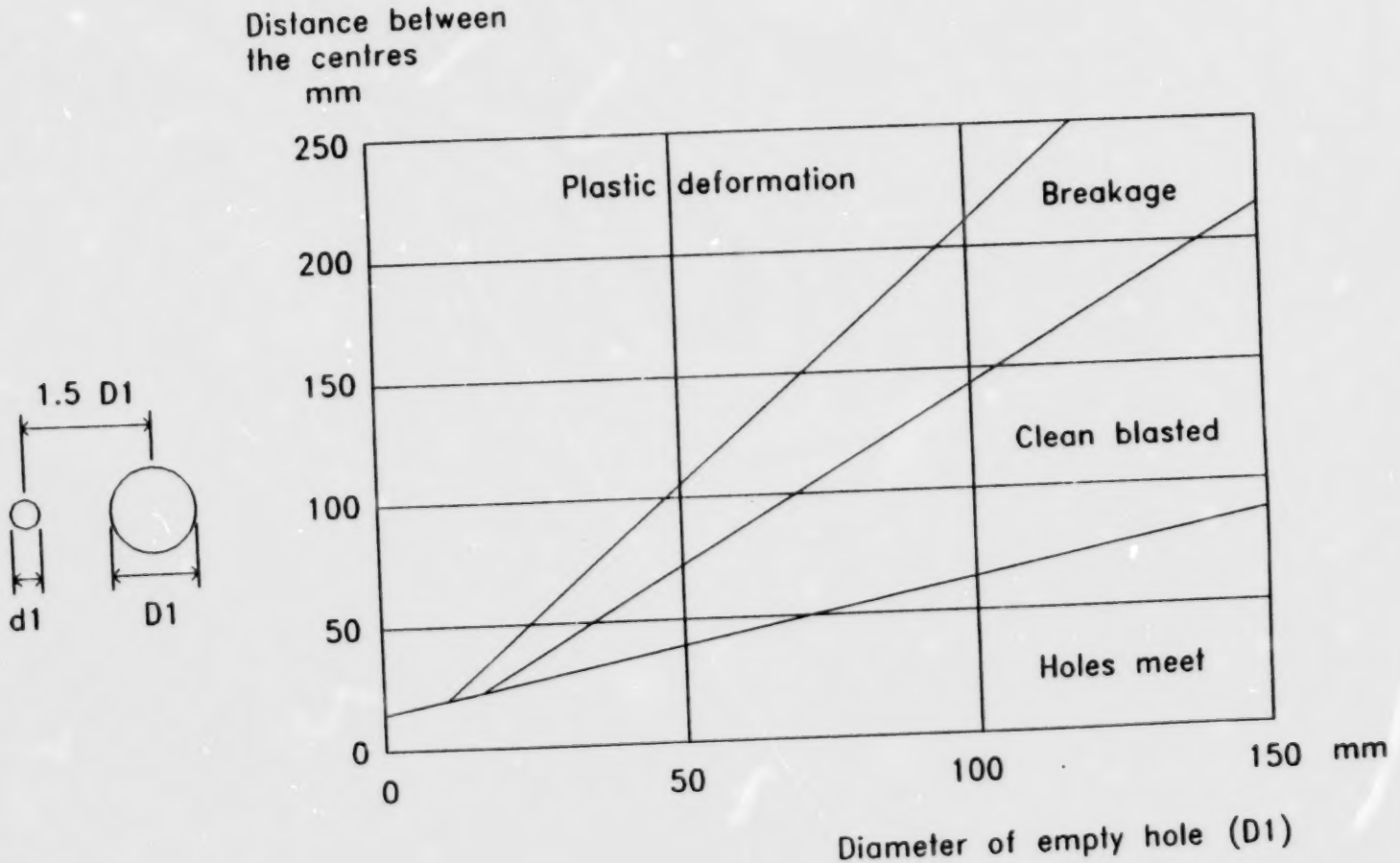


Figure 4 Effect of Diameter and Distance of Relief and Charged Holes on Blasting Efficiency

A common problem in development blasting is sympathetic detonation or flash over. A flash over is the initiation of a hole by another hole nearby. It is possible when the rock is heavily jointed: the explosion propagates through the cracks and reaches the adjacent hole, initiating it in the process. Another problem is the desensitization of the explosive. All explosives have a minimum explosion diameter. For a free flowing explosive like ANFO, this is a serious problem since the explosive is easily compressed. The shock wave can compress the ANFO to a diameter less than the critical diameter and cause a complete failure to detonate or a deflagration. (A deflagration is a low order detonation velocity which is too low to break the rock). Since the shock has a short duration the explosive will re-expand to its original size afterwards and if the caps are of the long delay type the explosive column may recover in time. For short delays it is more critical. Sympathetic detonation and desensitization mostly occur in the cut region and both can seriously impede blast performance.

Types of Cuts

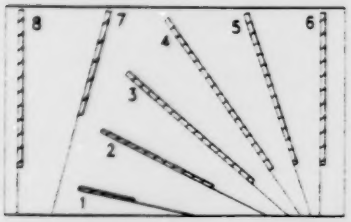
There are three main types of cuts: fan cut, V cut and parallel hole cut. Figure 5 illustrates the different kind of cuts. Langefors and Kihlström[3] provide a good review of these techniques.

In a fan cut design, the detonation sequence is chosen in such way as to give each blast hole a free breaking face. Many variations exist but they are all using the same principle. The face to be blasted has to be sufficiently wide in order to drill the angled holes.

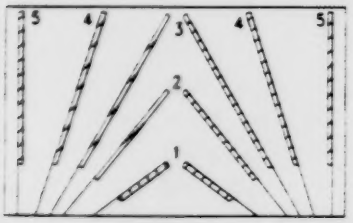
The V cut consists of opposed angled holes. The first set of holes create a small crater and the subsequent sets enlarge the crater. The cratering is enlarged until a full face is achieved. The drilling accuracy is important to the cut success and, as in the fan cut, it requires a wide face to permit the equipment to drill the angled holes. The advantage of this cut is that it requires less holes than the parallel hole cut.

Neither the fan cut nor the V cut can be implemented in most Canadian metal mines since the drift is not wide enough to permit the drilling of angled holes.

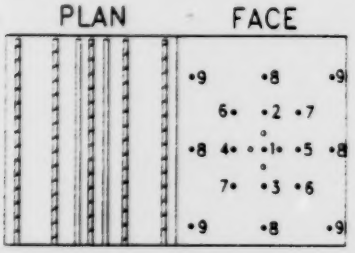
The parallel hole cut can be divided in two categories: burn cut and cylinder cut. In a burn cut, the cut is heavily charged and the rock is pulverized and "burned". The term burn is used because burn traces can be noticed on the rock after the blast. It must be noted that not all types of rock are suitable for a burn cut. In some type it is not possible to "burn" the rock; you then have to use a cylinder cut. The cylinder cut works with a lower explosive charge. The first blast hole breaks into the central relief holes. The second, breaks into the gap enlarged by the first hole and so on. The explosive gases carries the broken rock outside of the cut during the process. Since the amount of rock to break is small the charge needed is also relatively small. An extensive study from Langefors and Kihlström[3] for these two cut types showed that, in most cases, the cylinder cut was more efficient.



FAN CUT



V CUT



PARALLEL HOLES CUT

Figure 5 Section View of the Three Main Types of Cuts in Development Blasting

Drilling

Rock drilling technology has advanced rapidly since the 1960's. Figure 6 shows rock drilling methods advancement and use as of 1960. Hand held drilling using a jackleg is very labor intensive but still used in some operations today (mostly small vein mining). However, in the 70's, it was replaced by the development of pneumatic jumbos, giving one man capability of completing many hand held drill tasks. Rapidly, rising labor costs and physical demand were reduced.

Hydraulic drilling equipment advanced mechanization one step further with the increasing drilling rates and improved operator working conditions. Hydraulic drills are still in development today, becoming more efficient as well as more reliable and economical. The next step of drilling development is automation. The two largest international drilling equipment manufacturers, Atlas Copco and Tamrock, both supply automated versions of hydraulic jumbos, which provide computer aided rock drilling. The greatest advantage of computer aided rock drilling is the greater accuracy obtained which permits drilling sophisticated blasting patterns and minimize overbreak.

Drilling precision is an important factor to a successful round. For a small advance (6-8 ft), a good precision can be obtained with pneumatic or hydraulic drills. When the depth increases, hole deviation and small orientation errors become very important. This is most critical in the cut since the holes are close together and deviation can cause sympathetic detonation or desensitization. If the rock is not homogeneous, the bit will tend to deviate towards the soft rock. The driller must then decrease the pressure on the bit in order to decrease deviation.

At BMS-M almost all the jumbos used for development are of the three boom pneumatic type and are 15 years old and more. The operator has to be extremely careful when drilling, because with this type of machine, deviation can be quite high. The drilling rate is also 25-30% slower than with hydraulic jumbos, however, at BMS-M this does not significantly affect the rate of advance as rock bolting and mucking are the "bottlenecks" of the full round cycle.

Drilling Practice

When starting a new round, the drill operator must verify that there is no loose rock on the roof or on the face. He marks the face as follows: after cleaning the face, he paints every bootleg with white paint and draws a regular white grid which determines the cut area and the rest of the hole positions. As the law requires, he keeps the new hole location a minimum of 6" from any bootlegs. He then proceeds to the drilling. Each side boom drills the lifters, stoping and back holes. The middle boom is mainly used to drill the cut area and some stoping holes.

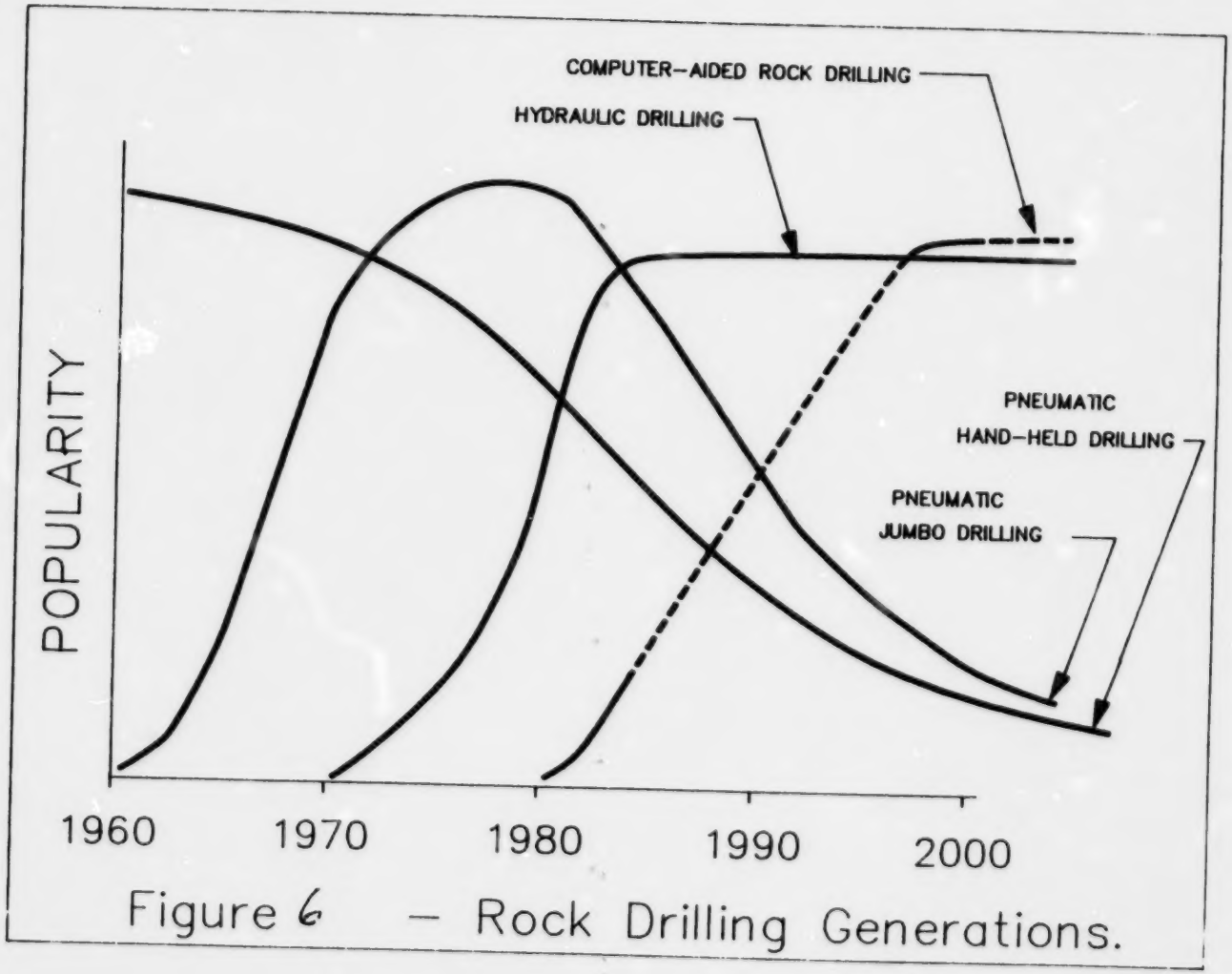


Figure 6 – Rock Drilling Generations.

During the drilling process, the operator has to change the worn bits from time to time. Some bits will wear faster than others and some will produce more deviation depending on the design and the material used. The steel rods also have to be changed from time to time. The steel rods can break or wear out to the point where they can't anchor the bit properly. They have to be changed or repaired when needed. A reamer bit is more likely to get stuck inside the hole. If this occurs, it must be extracted with a special tool or another hole must be drilled.

Once the drilling is completed, the round is loaded with ANFO and fired from surface. The driller will then evaluate the efficiency of rock fragmentation and the overbreak created. Since these two factors are in conflict, the objective is to find a balance between the two. If the wall and rock damage is important, it is possible to modify the pattern or use a decoupled explosive in the perimeter holes to reduce the damage.

1.2 PROJECT PLANNING

1.2.1 The Mine and The Test Area

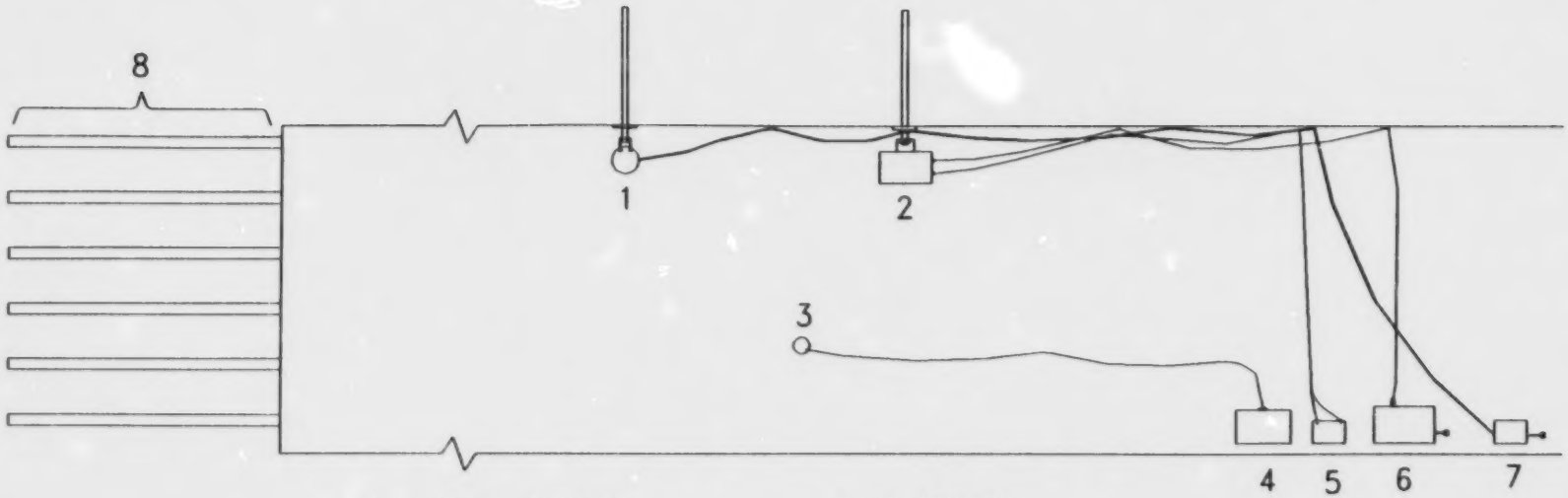
Brunswick Mines is situated in the north east of New Brunswick in Canada. Situated 25 km south of Bathurst, the mine produces zinc, lead and copper concentrates. The ore is massive sulphide (pyrite, pyrrhotite and galena) which grades from 7 to 20 % (Zn+Pb). The mining methods used are long hole and mechanized cut and fill. The test area was situated on the 1000m (3250') level #2 sub.

The ore body extends from surface to 1100 m below surface at a dip of 85 degrees. The geology is massive sulphide with no fault and few joint systems. The rock stress level is insignificant and no stress cracks were recorded in the test area. The material is brittle and usually easy to break with explosive. The geology varied from high pyrite (waste sulphide) to high galena and high pyrrhotite (high grade). Pyrite is present in all the grades. The geology present itself in vertical alternating veins of pyrite and ore (pyrrhotite and galena). The geologists evaluate the grades in situ and separate the areas into waste sulphide, low, medium and high grade. The main ore body is surrounded by a layer of massive pyrite. On the 1000m level the ore body is about 50 m wide.

The drift are usually 5.5 m (18') wide by 4 m (13') high. The cut has 3 or 4 reamed holes, 75 mm (3") diameter and 3 or 4 charged holes 37 mm (1.5") diameter. The round contains approximately 45 charged holes but this number can be as high as 52 for some mining crew. Figure 12 shows a typical round. All the hole are loaded with ANFO and detonated with nonel caps.

1.2.2 Instrumentation

The system chosen was as simple as possible to provide maximum flexibility. The geophones had to be cemented in the wall at least 24 hours before the blast to allow the cement to harden. Once completed, all that was required was to connect the geophone cable into the vibration recorder. When everything was connected, the team left for the surface and waited for the central blast before going back underground to gather up the equipment and evaluate the results of the blast (Fig. 7).



1. LAMP SYSTEM
2. CAMERA
3. GEOPHONE
4. VIBRATION RECORDER
5. BATTERY
6. VIDEO RECORDER
7. LAMP TIMER
8. ROUND

Figure 7 Instrumentation Set-up in a Heading

Video System

The video system consisted of two cameras (one of them was a spare) of very low light sensitivity (3 Lux). They were enclosed in a special shock mounted aluminum case with a LEXAN window in front to protect the cameras against flying rocks. There was one color camera and one black and white. The black and white camera was more sensitive to light and preferred for the tests (see Appendix #2). The cameras were powered by 12 volts batteries. Although, it was first planned to mount the batteries inside the case, the outside mounting was finally chosen because a longer recording time was possible.

The video signal was sent through a 400 foot coaxial cable to the recorder. The first 50 feet was protected against fly rocks by a high strength plastic shell. The recorder was enclosed in a carrying box and protected with foam. This protected the recorder from shock, vibration and dust. The images were recorded in Super VHS at speed of 60 images per second or an image every 16.6 ms.

The camera was secured to the back by screwing it to a rock bolt. The video and power cables were fixed to the roof and then run to the recorder and battery. Once the setup complete, a small monitor was used to verify the signal transmission and the camera alignment. The recorder was programmed to start and remain on for 40 minutes (this extended time provided insurance in the event of a blast delay; see Fig. 8).

Light System

The light system was designed in such way as to be easy to move, hang and maintain (Fig. 9). The light was provided by four slide projector lamp fed with a 220 volts line. They were mounted in special sockets in order to be replaced rapidly when required. The socket itself was screwed inside a stainless steel pipe (4 feet long and [illegible] diameter) in which rectangular holes were made and covered with LEXAN sections. Since the lamps generated tremendous heat and the LEXAN melts at 70^0C, a timer was used to shut the lamps after 30 seconds.

Vibration Monitoring System

The vibration monitoring system consisted of geophones, cables, recording machine and a computer. The geophones were cemented in a six foot deep hole in a rock wall. They were oriented towards the blast to pick up the vibrations from every hole of the round. The vibrations were translated into voltages by the geophone. This voltage was sent through a coaxial cable to the recorder and stored in its memory. After the blast the information was transferred to a computer disk and subsequently analyzed on surface.

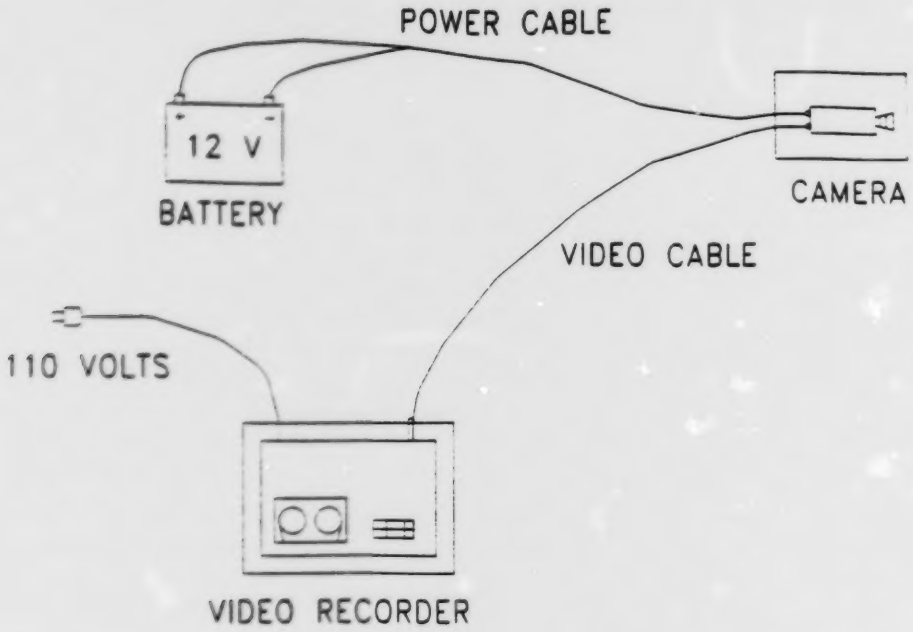


Figure 8 Recorder Set-up

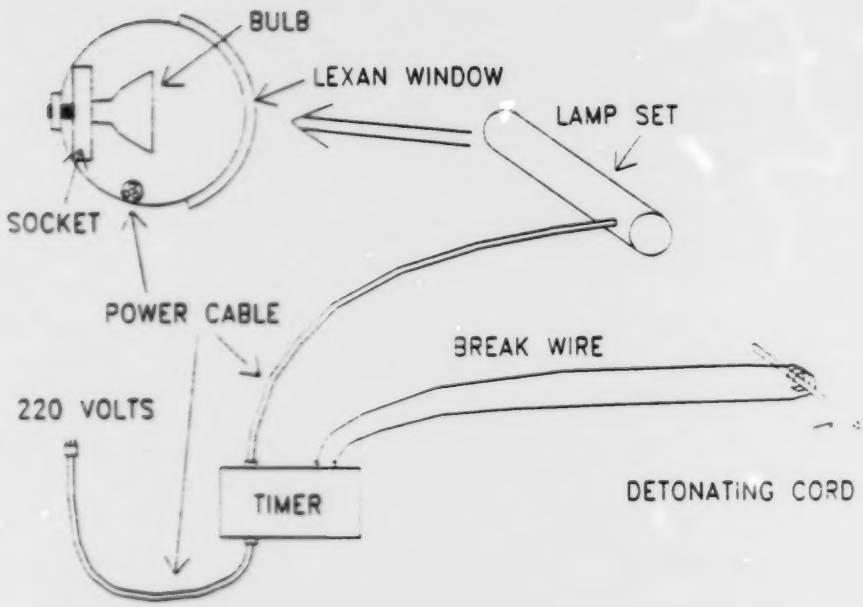


Figure 9 Lighting System Set-up

The vibration showed if all holes fired properly or else if some holes misfired or went off at a low order velocity. It also showed the accuracy of the caps and the occurrence of sympathetic detonation. To ensure proper detection, vibration gauges were installed on both sides of the blast. This eliminated any possibility of interpreting a low peak on the trace as an inefficient detonation, when it was due to the extended path of the vibration wave. All the vibration monitoring was carried out by JKMRC. They provided all the required material and manpower for the duration of the project and their field experience was a very valuable asset to the project.

1.2.3 Schedule

The field work started on August 16th and extended over a period of 12 weeks, finishing on November 10th. The tests were done alternately on day and night shift. This was required in order to follow the development crew schedule.

PHASE 2 - FIELD WORK

2.1 INITIAL PREPARATION

2.1.1 Project Timing

Members of the project team from NTC and JKMRC initially assembled in Montreal on 7th August 1989 to discuss and finalize project objectives.

The research group arrived at the BMS mine on 17th August, and after mine induction, initial discussions and equipment installation, Stage 1 monitoring commenced on 24th August.

The initial group on site consisted of the following personnel:

- NTC Project Supervisor (J.P. Saindon)
- JKMRC Project Co-ordinator (G.P. Chitombo)
- JKMRC Engineer (P.A. Walker)
- JKMRC Engineer (M. McKew)
- Noranda Engineer (Y. Dionne)

The project was run as a joint NTC/JKMRC project. Timing of the three stages are detailed in Table 1.

TABLE 1

Project Timing

Stage	Start Date	Finish Date	Weeks
Setting up	17/08/89	23/08/89	1
1	24/08/89	14/09/89	3
2	15/09/89	03/10/89	3
3	04/10/89	24/10/89	3

2.1.2 Objectives

Various methods of controlling secondary dust ignitions have been tried at Brunswick Mine with limited success. These have included the use of lime, watergel stemming and spray curtains. This project was devoted to reducing the risk of secondary explosions through improved blasting techniques.

Previous work at NTC suggested that sulphide dust clouds generated during a development round were possibly ignited by holes which either misfired or detonated poorly (low order detonations, LODs) allowing the ANFO column to burn, thus providing the source of the reactive dust ignition. The initial aim (Stage 1) of the field work was to identify the extent of such occurrences.

The second hypothesis submitted for testing was that the short period delays would be beneficial. It was thought that if all charge holes could be fired prior to the formation of a substantial reactive dust cloud (high dust concentration), then an ignition might be avoided or at least the effects minimized. The second part of the field work (Stage 2) was therefore designed to evaluate the potential of using short period delays.

After reviewing results from Stages 1 and 2, alternative objectives were developed because the initial ideas had shown limited potential. The third segment of the field testing (Stage 3) thus concentrated on development blasting testing from additional underground sites, and limited testing of various explosives.

2.1.3 Test Area and Crew

The 1000m #2 sublevel was under development and there was only one development crew in the north end. No cross shift or any other crew was in that section, which ensured that all the methods and practices were the same from blast to blast. Many headings were available on the level ensuring a continuous availability of blast. As stated before, the geology varied from massive pyrite to mixed pyrite, pyrrhotite and galena.

The mining crew was very familiar with the test section. Each crew member has about 15 years of experience in mining and they were highly skilled for the job:

- **Ron Jessulat:** Jumbo Operator, 17 years experience; Background in Industrial Engineering.
- **Frank Godin:** Loading, Blasting and Mucking, 16 years experience.
- **Cletus Morrisson:** Bolting; on the project from August to end September.
- **Donald Plourde:** Bolting; on the project from October to November.

Throughout the three months, the crew was highly committed to the project's success and involved in the planning and evaluation of the work.

2.2 BLAST AUDITING (STAGE 1) (August 16th to September 14th)

2.2.1 General

Stage 1 of this project was devoted to auditing current development blasting practices. Monitoring during this stage was limited to one crew (R. Jessulat, etc...) working on 2 sub north. Later, Stage 3 was used to gather data for other crews working on 2 sub south and 1 sub north.

The blasts were audited in the following categories:

Design Parameters
- Drilling
- Powder Factor
- Loading Practices

Blast Performance
- Timing
- Misfires
- Poor Detonations (Low Order Detonation)
- Fragmentation
- Advance
- Face Shape
- Over-break

Explosive Distribution

2.2.2 Monitored Blast Descriptions and Locations

A total of 13 development firings were monitored during Stage 1 and a further 4 during Stage 3. The rounds were designated as BMS01 to BMS13 (Stage 1) and BMS26 to BMS29 (Stage 3). The location of the rounds are identified on Figures 10 to 13.

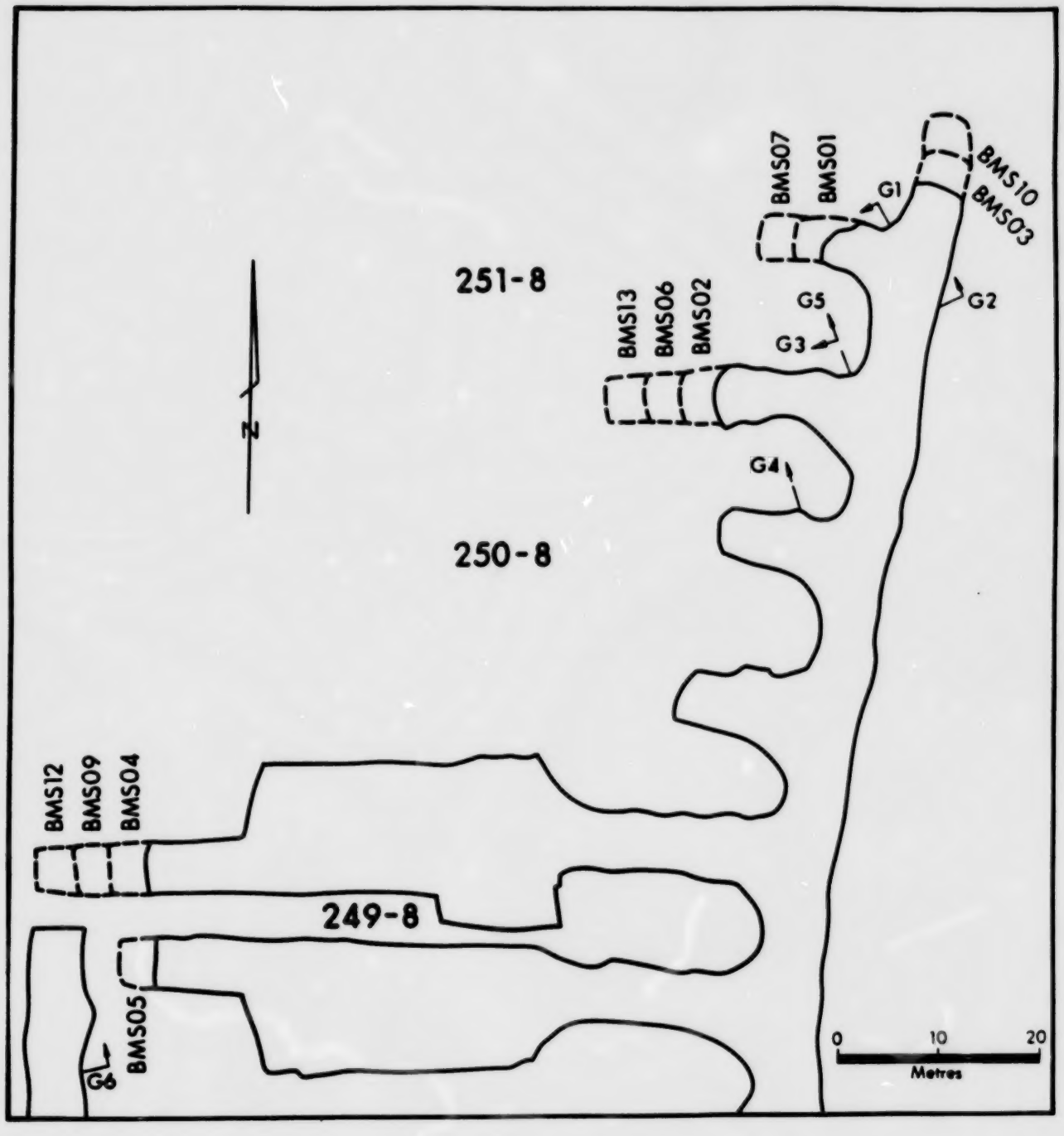


Figure 10 Approximate Location of Blasts: 2 Sub North

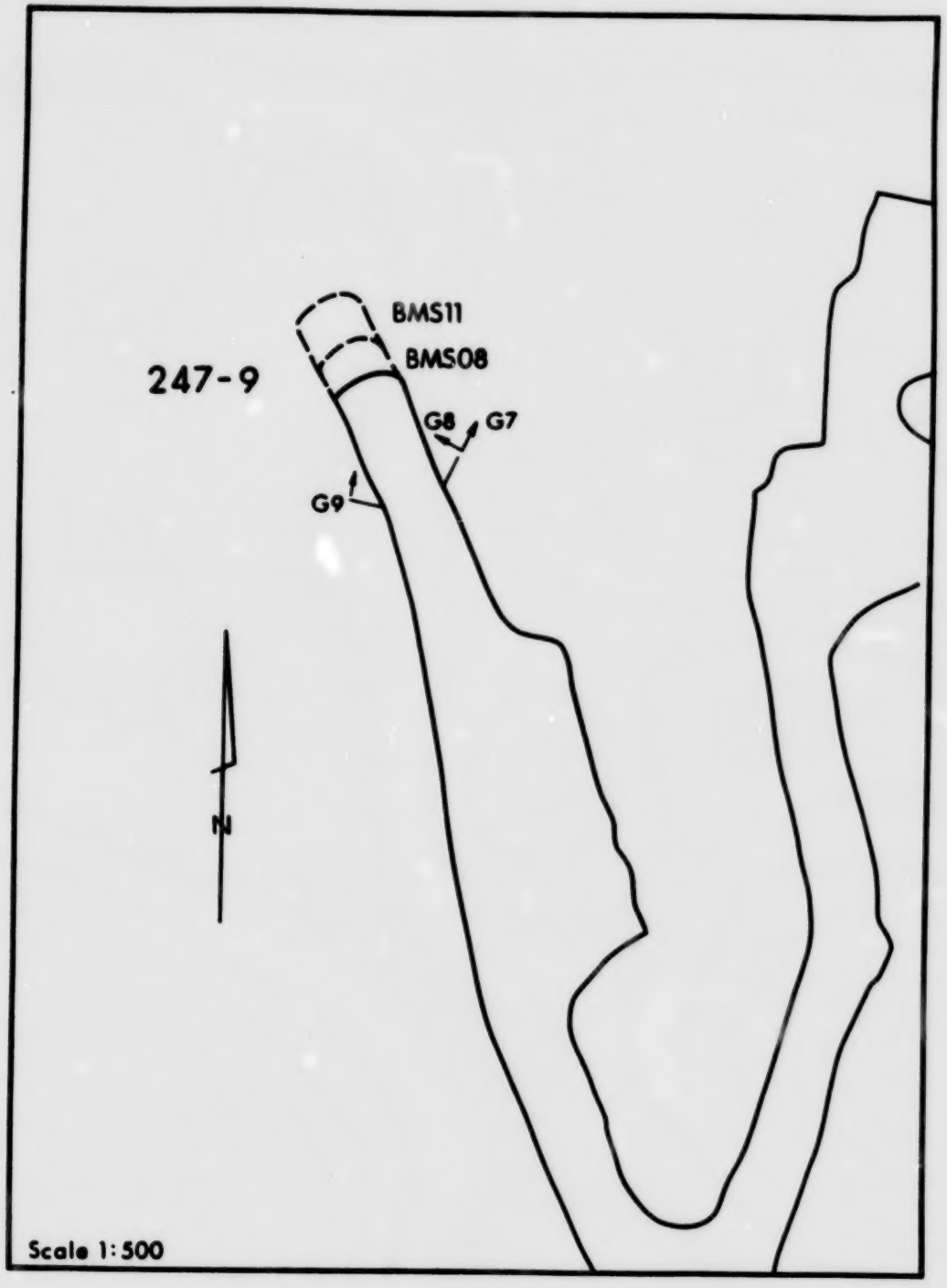


Figure 11 Approximate Location of Blasts: 2 Sub North Hangingwall

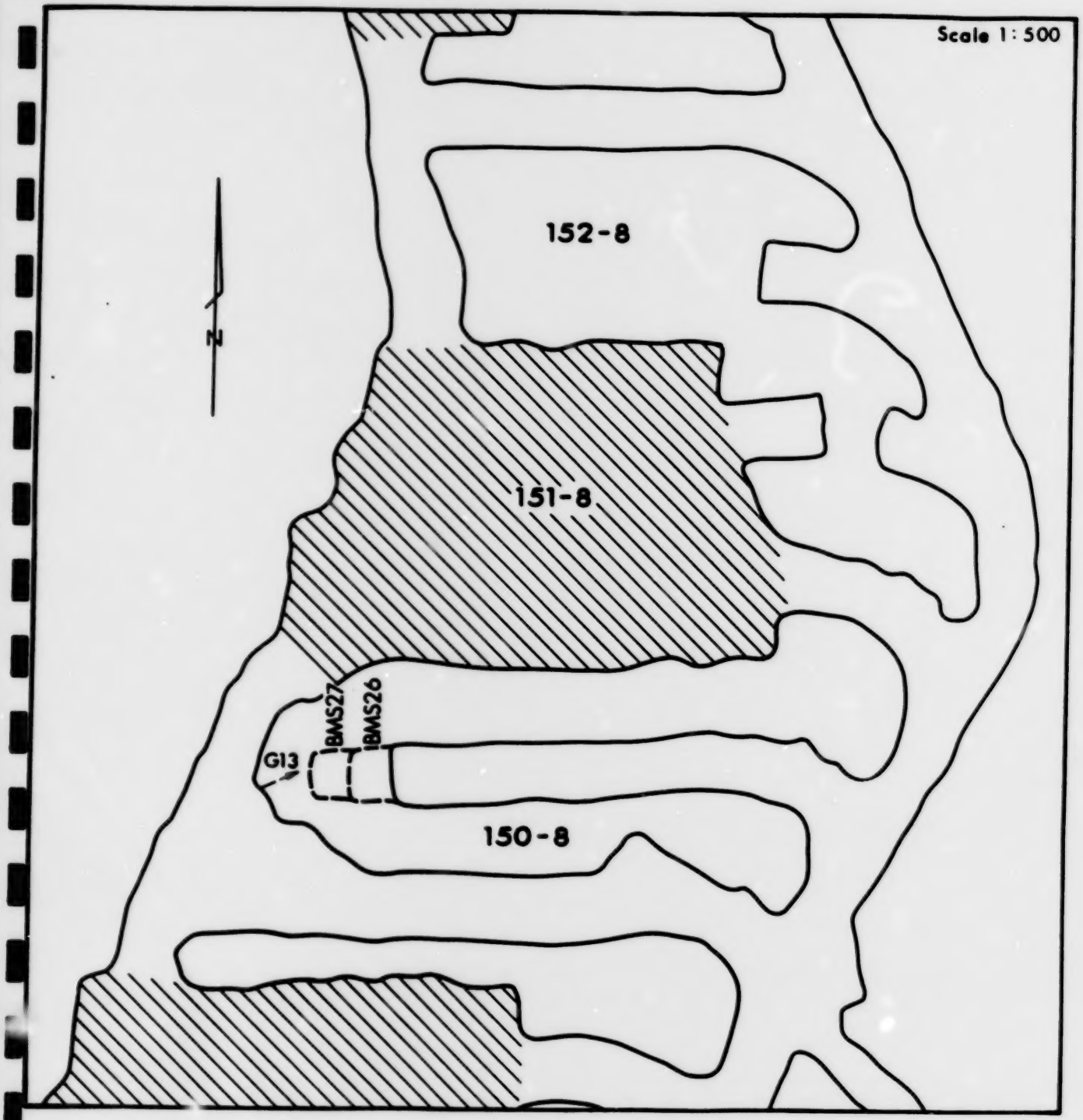


Figure 12 Approximate Location of Blasts: 1 Sub North

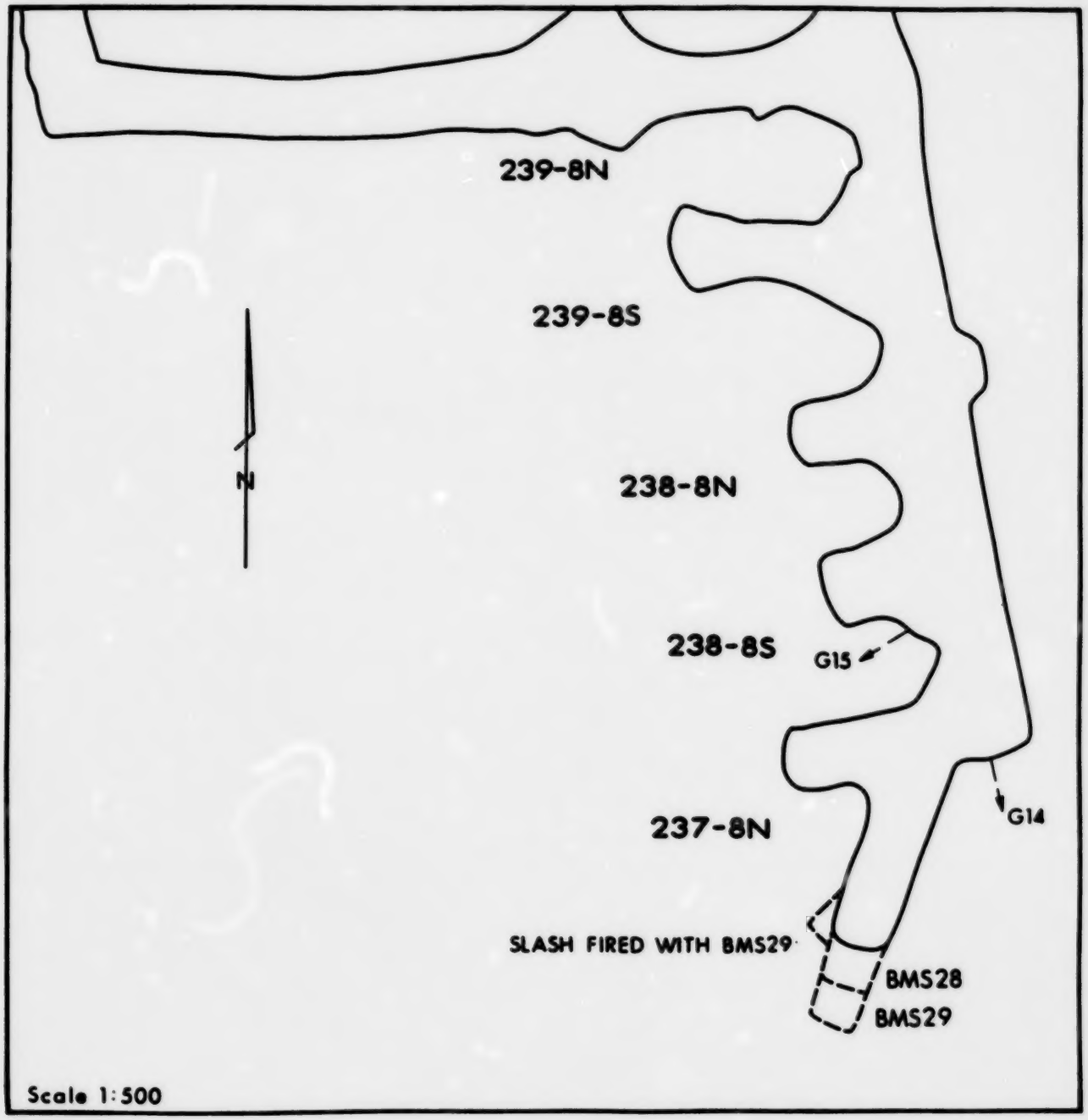


Figure 13 Approximate Location of Blasts: 2 Sub South

A description of the blasts monitored during Stage 1 are presented in Table 2. Table 3 contains a similar description for the blasts associated with Stage 3. During Stage 1 all of the possible rock/grain (rock fabric) combinations were monitored (refer Table 1), i.e.

a) Ore - Across Grain
b) Ore - Along Grain
c) Waste - Across Grain
d) Waste - Along Grain

Monitoring an equal number of rounds in all of the four combinations was not possible due to a lack of development headings, but sufficient experience was gained in each ground type for general conclusions to be drawn - refer to later comments. Not all combinations could be monitored during Stage 3 due to time constraints.

TABLE 2

Description of Blasts Monitored - Stage 1

Blast ID	Date	Location	Material	Direction
BMS01	24/08	251-8 North	Waste	Across grain
BMS02	25/08	251-8 South	Ore	Across grain
BMS03	29/08	F/W Drift North	Waste	With grain
BMS04	30/08	249-8 North	Ore	Across grain
BMS05	01/09	249-8 South	Ore	Across grain
BMS06	05/09	251-8 South	Ore	Across grain
BMS07	06/09	251-8 North	Waste	Across grain
BMS08	07/09	247-9 Drift	Ore	With grain
BMS09	08/09	249-8 North	Ore	Across grain
BMS10	11/09	F/W Drift North	Waste	Diag. across grain
BMS11	12/09	247-9 Drift	Ore	With grain
BMS12	13/09	249-8 North	Ore	Across grain
BMS13	14/09	251-8 South	Ore	Across grain

TABLE 3

Description of Blasts Monitored - Stage 3

Blast ID	Date	Location	Material	Direction
BMS26	12/10	1 Sub: 150-8 N	Ore	Across
BMS27	19/10	1 Sub: 150-8 N	Ore	Across
BMS28	19/10	2 Sub: F/W Drift S	Waste Sulphide	Along
BMS29	24/10	2 Sub: F/W Drift S	Waste Sulphide	Along

NOTE: Direction can either be along or across the inherent rock fabric.

2.2.3 Design Parameter

Drilling

Drilling details are contained in Table 4 below. Typical drill hole layouts are displayed as Figures 14 and 15 representing blasts BMS04 and BMS29.

Thirty-seven mm diameter charge holes were drilled with relief holes reamed out to 75 mm diameter. Drill steels 3.6 m long were utilized but due to set-up constraints generally only 3.5 m deep holes were achieved.

Some of the blasts monitored had additional "slash" holes drilled around their periphery - these are noted in Table 4. The extra slash holes were used to provide additional clearance on corners or to alter a drift direction.

If the additional slash holes are ignored, between 42 and 46 charge holes were being drilled to provide the advance required. One crew however was observed to drill 50 charge holes - refer to later comments on the poor performance of this blast. It is interesting to note that the number of holes and pattern used were independent of the rock/grain fabric combination.

TABLE 4

Drilling Details

Blast ID	Holes Drilled		Comments
	Charge	Relief	
BMS01	49	4	5 slash holes for corner
BMS02	49	4	4 slash holes on side
BMS03	44	4	
BMS04	44	4	
BMS05	44	4	
BMS06	45	4	
BMS07	46	4	4 slash holes on side
BMS08	46	3	
BMS09	42	3	
BMS10	48	4	4 slash holes on side
BMS11	43	3	
BMS12	46	3	
BMS13	43	3	
BMS26	50	3	
BMS27	45	3	
BMS28	43	3	
BMS29	44	3	

Powder Factor and Loading Practices

For blasts with between 42 and 46 charged holes the average powder factor for the development round varies from 2.0 kg/m3 to 2.2 kg/m3. Note this calculation allows for ANFO loaded into the drill holes only, (i.e. the remainder of bags discarded onto the floor is ignored).

All holes charged during this phase of the testing were pneumatically loaded with ANFO. The assumed density for compressed air loading is 0.98 g/cm^3.

Long period Nonel initiators were standard across the development crews monitored. They were loaded as near as practical to the bottom of the hole with the end of the cap pointing towards the end of the hole, i.e. no bend occurred in the Nonel tube. Small red Deta-prime boosters were tested during the trials (inserted over the Nonel caps) to verify if the added energy would provide a better cut performance. No significant increase in blast hole initiation reliability was recorded. Typical initiation sequences (delay numbers) are shown on Figures 14 and 15.

Differences between crews occurred with respect to stemming practices. The first 13 blasts monitored (BMS01 - BMS13) were loaded by the same crew (R. Jessulat, etc...). This crew did not use any stemming leaving approximately 300 mm collars for the cutholes and 600 mm for other holes.

Other crews monitored, completed every hole with an inert gelled water stemming cartridge. The holes were charged to within 300 mm to 500 mm of the collar and a stemming cartridge inserted. Several cartridges finished partly out of the holes. The cartridges were not tamped and some were ruptured during handling.

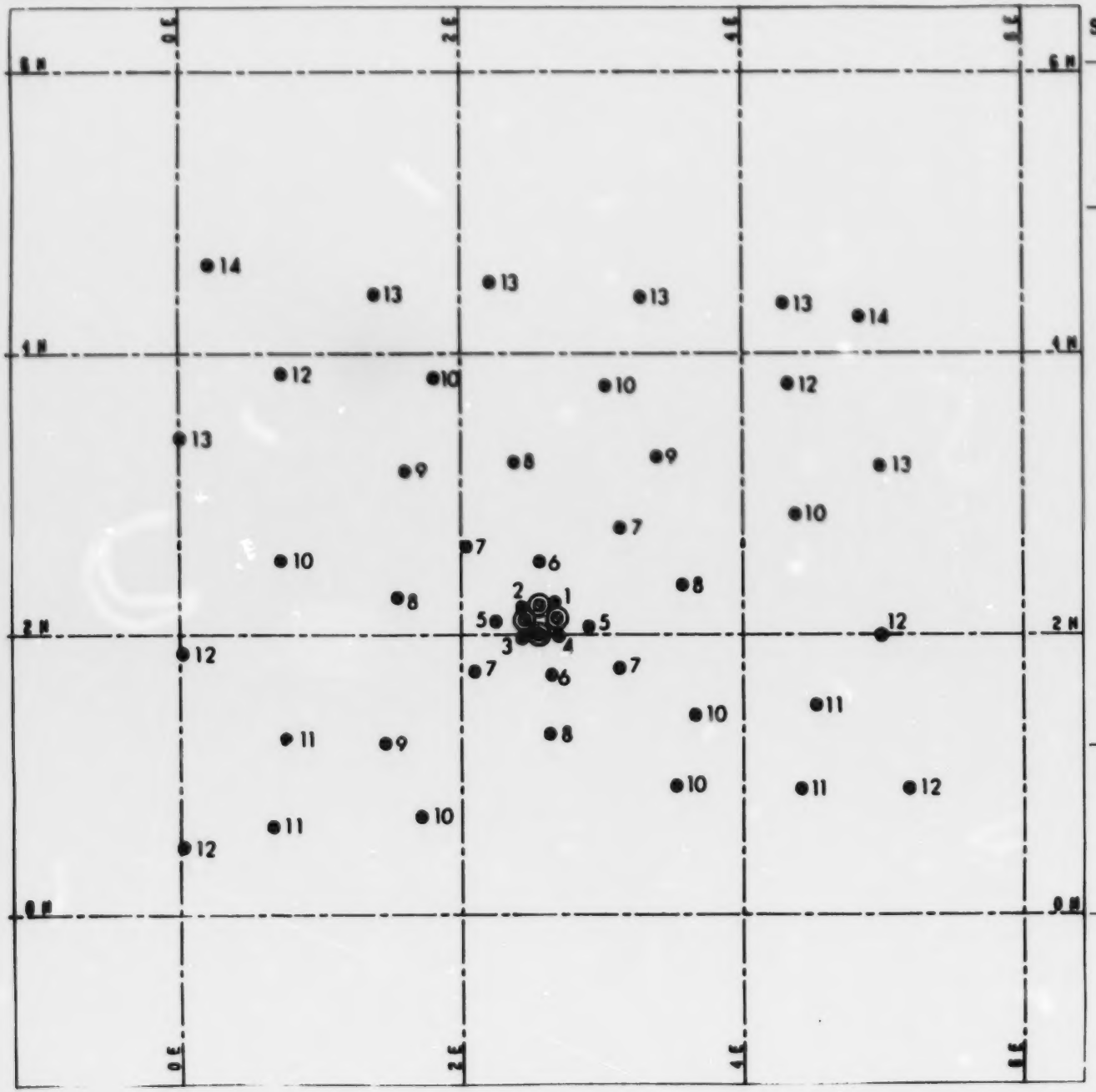
2.2.4 Blast Performance

Timing

The vibration traces recorded were analyzed to provide timing information for the holes which detonated. This information is presented in Table 5 below. The table combines information from both Stage 1 and Stage 3.

Inspection of the table shows the delays do generally detonate close to their nominal time, with minimal risk of delay overlap and sequence reversal.

The statistics in Table 5 have been generated assuming the scatter distributions to be normal. Although it is not considered necessary for this project, a detailed analysis would be required to prove this assumption.



Scale: 1:40 16-Nov-1999

BMS#4 : 30/08/89

Design: 249-8 Stope North
Plot of: Holes
Prepared by: JKMRC

Figure 14 Drill Hole Layout: BMS04

KEY TO ANNOTATION

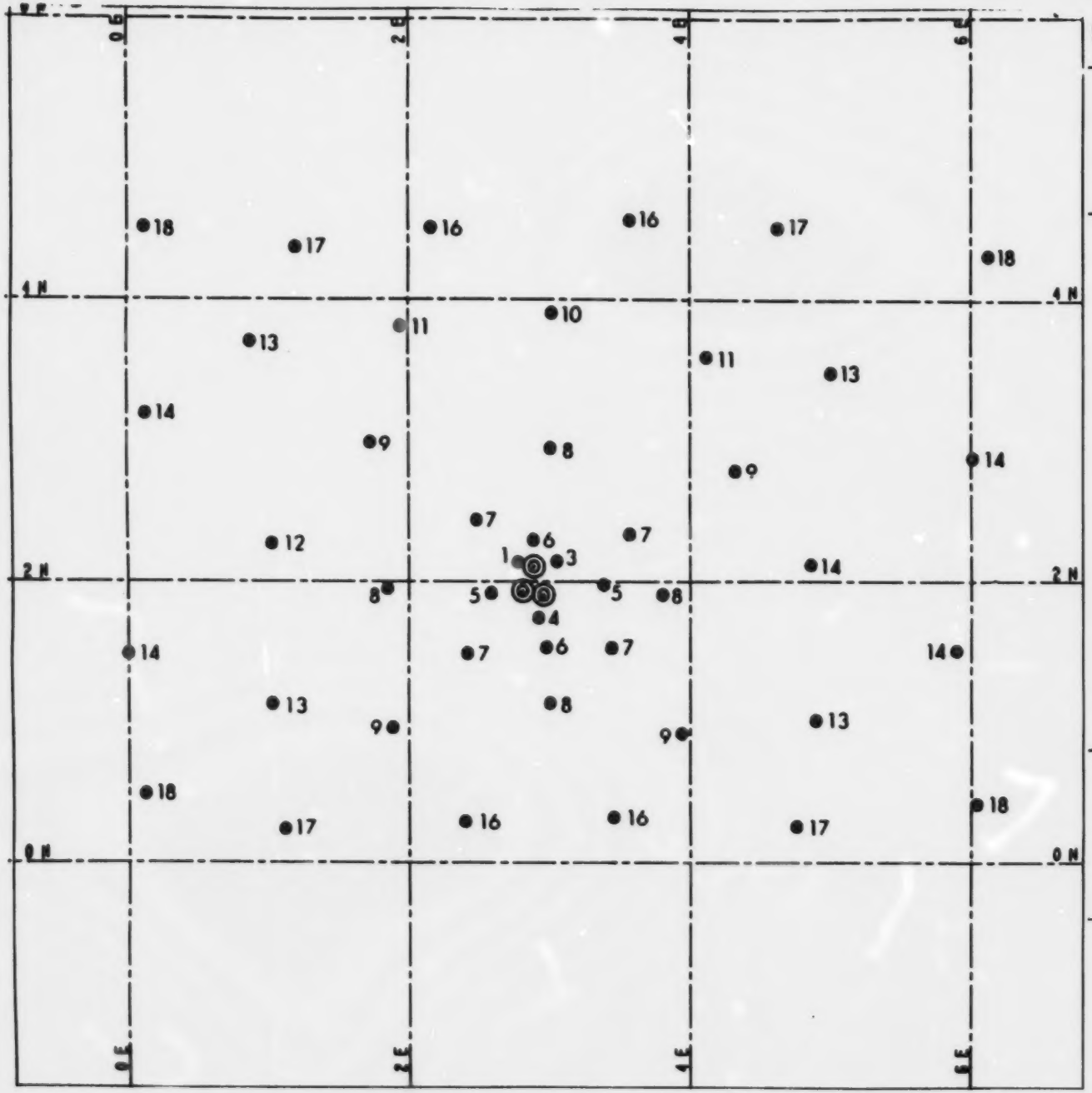
● Delay Number

BMS#4.GEN
BMS#4.FEA
BMS#4.HOL
BMS#4.EXP
BMS#4.TIE

JKMRC

3x3 V1.2

Ph. 61-7-[illegible] Fax. 61-7-[illegible]



Scale: 1:40 16-Nov-1989

BMS#29 : 24/10/89

Design: 3250/2 Sub F/W Drift South
Plot of: Holes
Prepared by: JKMRC

Figure 15 Drill Hole Layout: BMS29

KEY TO ANNOTATION

● Delay Number

BMS#29.GEN
BMS#29.FEA
BMS#29.HOL
BMS#29.EXP
BMS#29.TIE

JKMRC

3x3 V1.2

Ph. 61-7-870-0888 Fax. 61-7-870-5776

TABLE 5

Delay Scatter Analysis

Delay No. (Long Period)	Nominal Time (ms)	Statistics Generated from Monitoring				
		Samples	Mean (ms)	Std.Dev. (%)**	Min. (ms)	Max. (ms)
1	490	17	480	7	421	589
2	800	14	784	7	606	888
3	1125	17	1092	14	**489	1181
4	1400	18	1423	2	1362	1482
5	1675	24	1716	2	1652	1776
6	1950	34	1965	4	1774	2173
7	2275	51	2297	3	2114	2436
8	2650	54	2696	3	2412	2877
9	3050	61	3110	3	2922	3327
10	3450	73	3523	2	3269	3697
11	3900	75	3931	2	3711	4190
12	4350	71	4465	3	3866	4688
13	4850	19	4938	3	4568	5369
14	5350	42	5456	1	5312	5731
15	5900	31	6235	2	5901	6416
16	6550	11	6979	2	6610	7150
17	7250	10	7670	3	7283	8018
18	8050	6	8634	2	8324	8845

NOTE:

* Standard Deviation is quoted as a percentage of the mean.

** One low result was recorded. Possibly the hole detonated sympathetically or due to a flash over.

Misfires Recorded

Misfires are identified by missing vibration packets on the recorded traces. Results from analyses are displayed in Tables 6 and 7 for Stage 1 and Stage 3 respectively.

TABLE 6

Results of Vibration Analysis - Stage 1

Blast ID	Misfires			Poor Detonations		
	Cut	Easer	Perim	Cut	Easer	Perim
BMS01	2	2	1			1
BMS02						
BMS03	1	1	2			1
BMS04		1		1		
BMS05					1	
BMS06		2	2			
BMS07			1			
BMS08			1	2		
BMS09	1		1			
BMS10		3				
BMS11						
BMS12			2			
BMS13		1	2			
TOTALS	4	10	12	3	1	2

TABLE 7

Results of Vibration Analysis - Stage 3

Blast ID	Misfires			Poor Detonations		
	Cut	Easer	Perim	Cut	Easer	Perim
BMS26		2	1		3	2
BMS27			2	1	1	2
BMS28						
BMS29		2	1			1
TOTALS		4	4	1	4	5

The tables for Stage 1 and Stage 3 have been kept separate to note any differences between the crews monitored.

To further consider the results obtained, summaries by delay numbers are presented in Tables 8 and 9 below.

The planned delay numbers in these tables do not include those for blasts where vibration records were unavailable i.e. BMS02, BMS11 and BMS 28. For the purpose of this report the face has been roughly divided into three zones (Table 8 and 9) i.e.:

1) Cut Holes: The central four or five holes around the relief (uncharged) holes.

2) Easer Holes: The main body of the shot from the cut out to the perimeter holes. Included in this group of holes are the helpers and diamond holes.

3) Perimeter Holes: The outside one or two rows of holes, mainly round the back and walls.

TABLE 8

Vibration Results Summarized by Delay Number - Stage 1

Delay No.	Planned	Misfire	Deflagrate	Face Zone
1	11			Cut
2	11	1	1	Cut
3	11		1	Cut
4	12	2	1	Cut
5	1	1		Cut
5	20	4		Easer
6	24	1		Easer
7	42	3		Easer
8	44	2		Easer
9	44			Easer
10	63			Easer
11	70	4		Perimeter
12	72	5		Perimeter
13	14			Perimeter
14	36	2		Perimeter
15	22	1	2	Perimeter
TOTALS	497	26	5	

TABLE 9

Vibration Results Summarized by Delay Number - Stage 3

Delay No.	Planned	Misfire	Deflagrate	Face Zone
1	3			Cut
2	2			Cut
3	3		1	Cut
4	1			Cut
4	4	1		Easer
5	6		1	Easer
6	10		1	Easer
7	12	2		Easer
8	12	1	1	Easer
9	16		1	Easer
10	11	1		Perimeter
11	8			Perimeter
12	14	2	2	Perimeter
13	4			Perimeter
14	5		1	Perimeter
15	6		2	Perimeter
16	8	1		Perimeter
17	8			Perimeter
18	6			Perimeter
TOTALS	139	8	10	

During Stage 1, the majority of misfires occurred in the perimeter holes on delay numbers 11 and 12. A significant number of helpers (No. 5 delay) also misfired.

The results suggest that the helpers are too close to the cut holes and are being desensitized, but as discussed in more detail later (Section 4.0) these helpers are used as insurance. If the cut holes all fire then the close helpers are not required and can misfire without causing any problems. However, if some of the cut holes do not perform these helpers are needed to extract material from the cut area.

To put the results in their proper perspective, only 26 misfires were noted out of 497 planned detonations. This represents 5% of the holes or approximately 2 misfires per blast.

As these misfires appeared to be random in nature, major efforts would have been required to reduce this number, a task considered outside the scope of this project. It is not considered realistic (or necessary) to eliminate all misfires.

These results were confirmed with the monitoring of other crews in Stage 3. A total of 8 misfires were recorded from 139 planned detonations, representing 6%. The misfires for these crews were also distributed randomly across the shot.

Poor Detonation

The recorded poor detonations are displayed within Tables 6 to 9 above.

Many of the blasts monitored as part of Stage 1 could not be analyzed for poor detonations in holes outside of the cut with any degree of confidence. The total number of poor detonations detected is thus a conservative estimate. As with misfires, the number of deflagrating holes in Stage 1 represents a small percentage (1%) of the total holes fired.

In Stage 3 however the percentage increased to 7% but this was strongly influenced by the shot containing 50 drill holes (BMS26) - refer comments below.

The majority of poor detonations recorded in Stage 1 occurred in that cut. This suggests that the cut holes are too close together which was confirmed during Stage 3 where cut holes were further apart and fewer deflagrations were recorded.

Blast BMS26 (Stage 3) recorded the most number of deflagrations of any blast. This blast contained 50 charged holes (refer Table 4) and hence supports the theory that over-charging leads to increased problems within a shot.

Fragmentation and Advance

Fragmentation very rarely causes any concern with development blasting and no exception was found here. The muck was well fragmented and formed a pile close to the face suitable for removal.

Fragmentation photographs were taken from earlier blasts for subsequent digitizing and analysis as part of ongoing JKMRC research efforts. This topic is further discussed as part of Section 4.0 where levels of dust generated and rock trajectories are compared with those from the short period delays.

The advance achieved for each blast is listed in Table 10 below. As shown, acceptable meterage was pulled by each round, irrespective of the rock/grain combination being blasted. In general no bootlegs were left from cut holes, 100 mm - 200 mm from easer holes and up to 400 mm from perimeter holes.

TABLE 10

Development Advance Achieved per Round

Blast ID	Advance (m)	(%)
BMS01	3.3	94
BMS02	3.5	100
BMS03	N.A.	---
BMS04	3.4	97
BMS05	N.A.	---
BMS06	3.4	97
BMS07	3.6	100
BMS08	N.A.	---
BMS09	3.2	91
BMS10	N.A.	---
BMS11	3.4	97
BMS12	3.5	100
BMS13	3.5	100
BMS26	3.5	100
BMS27	3.5	100
BMS28	N.A.	---
BMS29	3.5	100

NOTES: N.A. - Not Available

Face Shape

No major problems with the shape of the face were encountered during the monitoring. On 1 sub (150-8 north access x-cut) several unfragmented lumps remained around the edges of the face from previous rounds but the shape did not deteriorate with subsequent blasts.

In general the new cut was drilled into approximately the same area as the previous one, slightly offset to avoid drilling into butts.

Over-Break

In general no problems with over-break were noted. The only area of minor concern was the footwall drift heading north on 2 sub. Over-break of the order of 500 mm was noted for this heading. The heading was passing through a dyke and over-break is thus attributed to the unique geology of the area.

2.2.5 Explosive Distribution

The JKMRC has recently developed a blast design software applicable to both underground and surface operations. The PC based software is designated as "3x3" because it applies editing, analysis and reporting to drilling, loading and sequencing.

A two-dimensional prototype 3x3D for development round applications was used in the Brunswick exercise. The program features used were the Three Dimensional Explosive Distribution Analysis which is built into the 2-D prototype and the Sequence Animation and Timing Variation.

It is possible to calculate a three dimensional explosive distribution in any horizontal plane or vertical section running north-south or east-west. The section or plan mode is selected from the data entry form which also specifies a depth and the range for the contouring values.

The analysis uses a tessellation algorithm to calculate a contour with ever increasing resolution, until the screen's maximum resolution is reached. Often a result is evident well before this point, and the analysis can be terminated early. Also, the calculation only uses explosive in selected holes. By selecting only holes in a typical section of the blast it is possible to obtain a representative answer in a much shorter time.

Sequence Animation and Timing Variation

Sequence animation and timing variation simulates the manner in which a delay is detonated and in turn which further delays or decks of explosive are detonated from it. The analysis begins at one hole and proceeds until no delay elements are active. An active delay is one which has been ignited but has not yet waited out its delay time. Each simulation uses a new and random set of delay scatter based on normal probability statistics, nominal delay time and the standard deviation of the delay.

Overall the designs used at Brunswick exhibited an acceptable explosive distribution. Photograph 1 is the calculated explosive distribution of one of the designs. High powder factors, and therefore concentrations, are expected in the cut area.

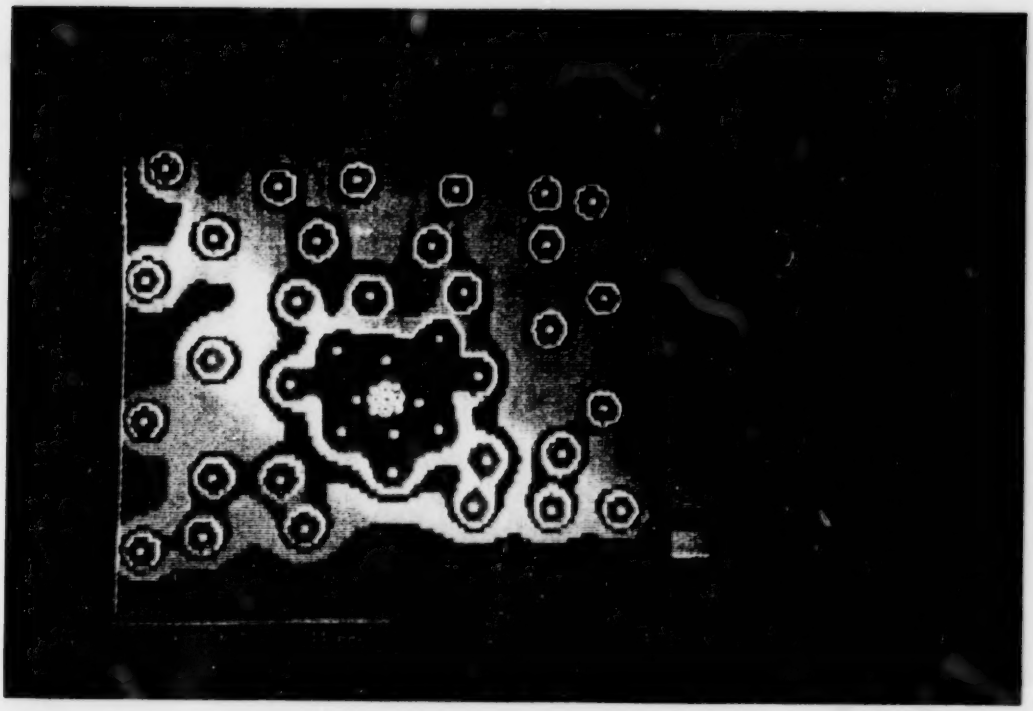
2.2.6 Summary

The different crews monitored were comparable with respect to development blast practices. Differences did however occur with respect to the use of inert stemming - refer comments above.

Approximately 44 charge holes were drilled to provide acceptable advance irrespective of the rock/grain combination.

Timing data extracted from the vibration traces showed the long period Nonel delays being used to be reasonably accurate with little risk of out of sequence detonations.

Misfires and poor detonations recorded (for Stage 1 and Stage 3 combined) represented a small proportion (8%) of the holes drilled. Reduction of these misfires and deflagrations would be a major exercise due to their random occurrences, with total elimination not considered a realistic goal.



Photograph 1 Calculated Explosive Distribution from 3x3D

To improve the round currently being drilled, attention should initially be directed to the cut. If the cut can be redesigned to ensure correct detonation of all holes then the close helpers currently being drilled as insurance could be deleted.

Again, if the cut can be repositioned on the face (slightly higher in R. Jessulat's case) it may be possible to eliminate a couple of holes from the face. This was demonstrated in BMS09 where only 42 charge holes were required.

2.3 TIMING (STAGE 2)

2.3.1 General

The main objective guiding the experiments conducted during Stage 2 was the reduction of timing and blast sequencing.

As a step towards reducing the timing of a development round, initial experiments consisted of cut, helper and diamond holes only. Later, this was further reduced to only the inner cut holes.

A series of 9 trials was monitored incorporating short period delays (25ms) supplied by ETI, starting with the fastest possible option and working back towards a reliable design. These experiments were terminated following a significant sulphide dust explosion and hence a satisfactory design using short period delays was not achieved.

2.3.2 Experiment Descriptions and Locations

The 9 trials monitored are described in Table 11 and shown on Figures 16 to 18.

Experiment BMS19 was used to confirm geophone response and analysis techniques being used. This experiment was reported in a separate memo issued to NTC on 16/10/89.

Experiment BMS20 consisted of three cuts fired together, separated by delay elements. The three cuts are referred to as BMS20-1, BMS20-2 and BMS20-3.

Blasts BMS15 and BMS21 were control shots using standard long period delays, for comparison with the short period delay experiments to avoid misinterpretations. BMS15 consisted of 12 charge holes whereas BMS21 contained 4 charged cut holes only.

TABLE 11

Description of Blasts Monitored - Stage 2

Blast ID	Date	Location	Material	Direction
BMS14	15/09	249-8 North	Ore	With grain
BMS15	18/09	247-9 Drift	Ore	With grain
BMS16	19/09	249-8 North	Ore	Across grain
BMS17	21/09	249-8 North	Ore	With grain
BMS18	22/09	249-8 North	Ore	With grain
BMS20-1	27/09	249-8 North	Ore	With grain
BMS20-2	27/09	249-8 North	Ore	With grain
BMS20-3	27/09	249-8 North	Ore	With grain
BMS21	03/10	249-8 North	Ore	With grain

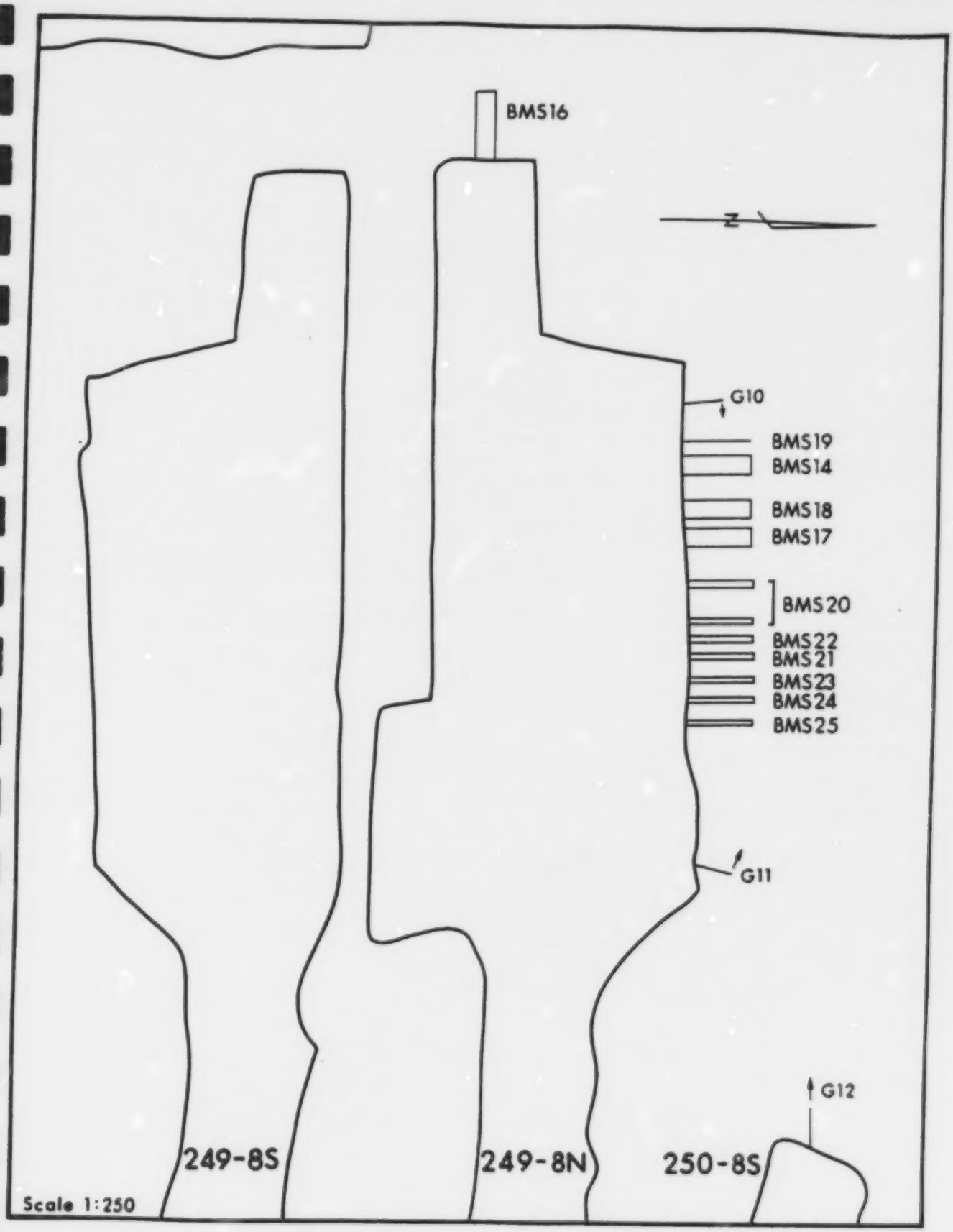


Figure 16 Location of Stage 2 Experiments: 2 Sub North

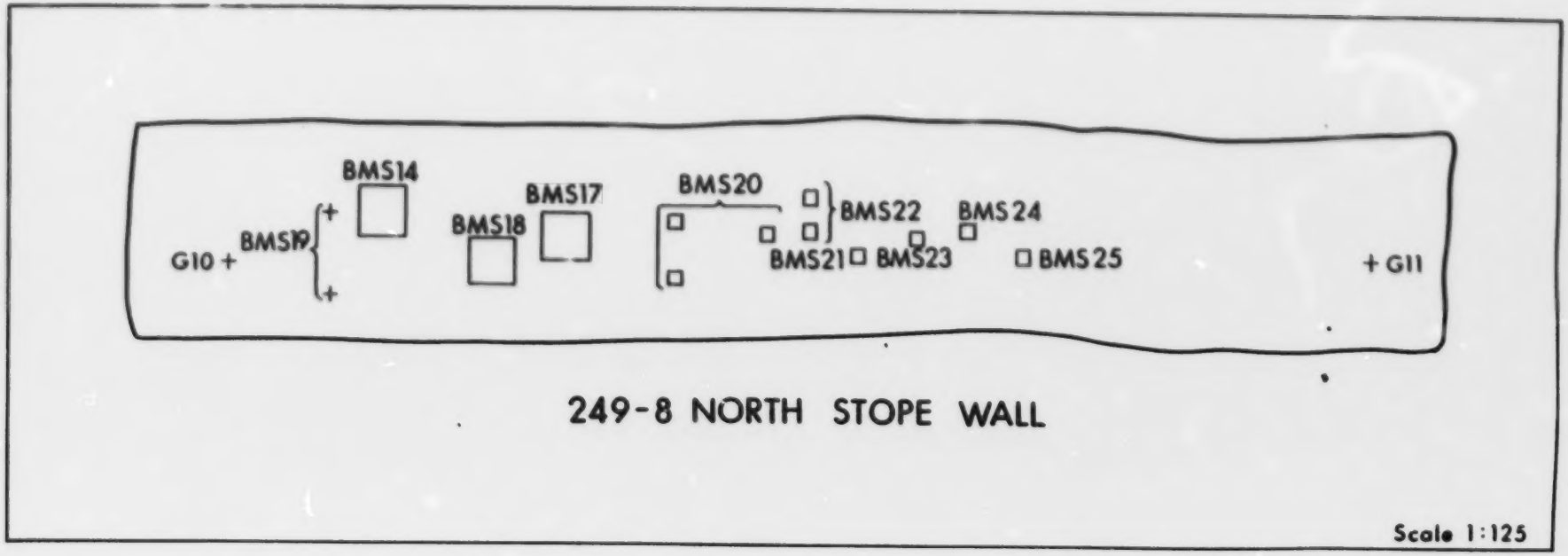


Figure 17 Cut Experiments: 249-8 North Stope Wall

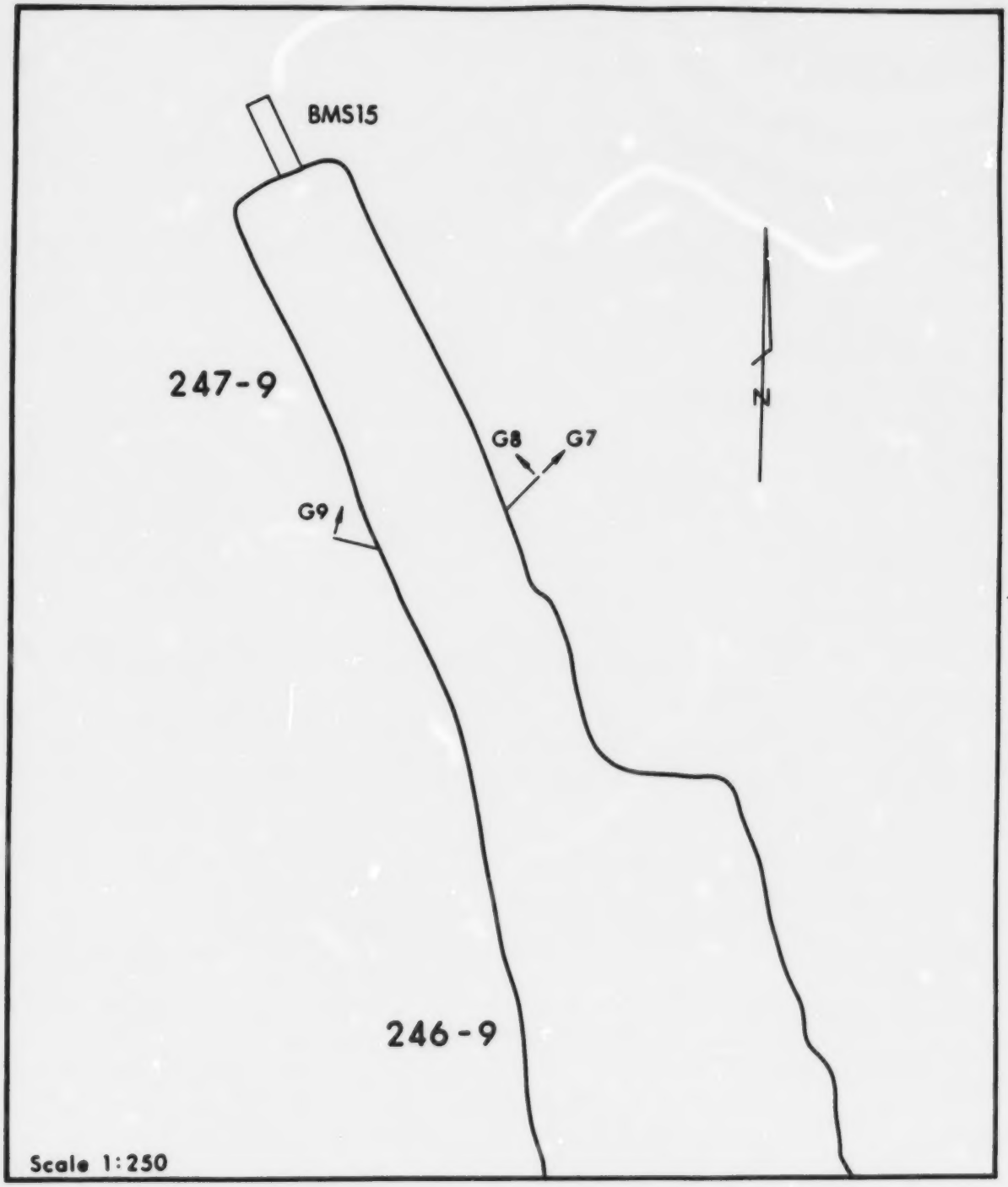


Figure 18 Location of Experiment BMS15

2.3.3 Results

Experiments trialled during this segment of monitoring were designed after reviewing results from the previous trial. Discussion is therefore best presented in chronological order, drawing on the major results as presented in Tables 12 and 13 below.

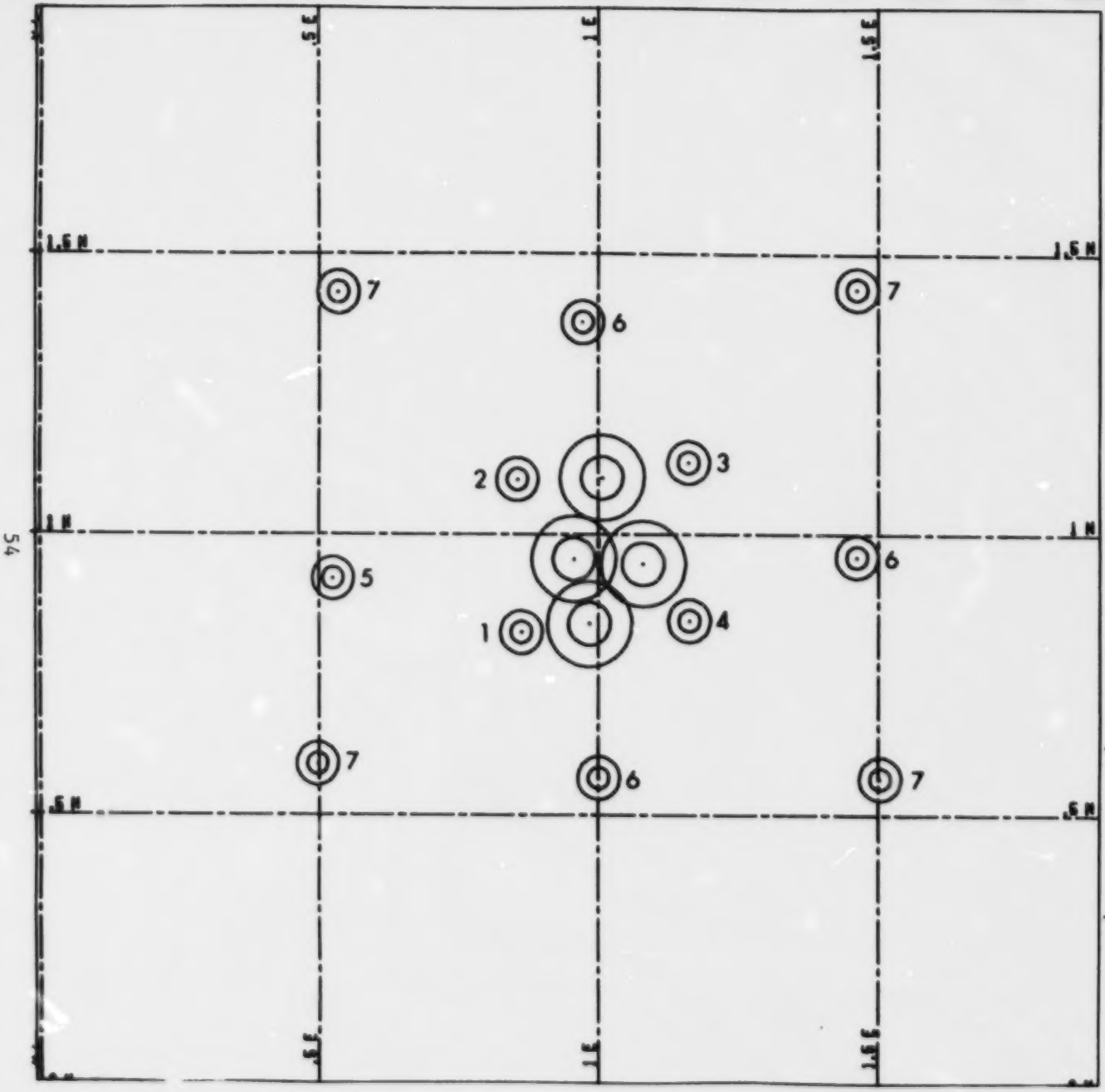
To minimize the number of changes between each trial it was decided to maintain the standard cut pattern used by R. Jessulat - see Figure 19. However one exception was tried, the corkscrew cut. Refer to comments below (BMS17).

TABLE 12

Results of Vibration Analysis

Blast ID	Misfires			Poor Detonations			Notes
	Cut	Helpers	Peri.	Cut	Helpers	Peri.	
BMS14			2	1			
BMS15		2					
BMS16				2			
BMS17	--	--	N.D.	--	--	N.D.	a
BMS18	--	--	--	--	--	--	a
BMS20-1	--	N.D.	N.D.	--	N.D.	N.D.	a
BMS20-2	--	N.D.	N.D.	--	N.D.	N.D.	a
BMS20-3	--	N.D.	N.D.	--	N.D.	N.D.	a
BMS21	--	N.D.	N.D.	--	N.D.	N.D.	

NOTES: a - Record was lost due to equipment failure
N.D. - Not drilled



Scale: 1:10 16-Nov-1989

BMS#14 : 15/09/89

Design: 249-8 Stope North
Plot of: Holes
Prepared by: JKMRC

Figure 19 Drill Hole Layout: BMS14

KEY TO ANNOTATION

◎ Delay Number

BMS#14.GEN
BMS#14.FEA
BMS#14.HOL
BMS#14.EXP
BMS#14.TIE

JKMRC
3x3 V1.2
Ph. 61-7-870-0000 Fax. 61-7-870-5775

TABLE 13

Depth of Pull - Trial Cuts

Blast ID	Depth of Pull (m)			Delay Speed	Notes
	Cut	Helpers	Perimeter		
BMS14	3.55	2.90	1.70	Short	
BMS15	3.45	3.45	3.30	Long	Control cut
BMS16	3.45	3.25	2.65	Short	
BMS17	2.60	1.60	N.D.	Short	Corkscrew cut
BMS18	3.55	3.22	2.22	Short	
BMS20-1	0.65	N.D.	N.D.	Short	
BMS20-2	0.40	N.D.	N.D.	Short	
BMS20-3	1.30	N.D.	N.D.	Short	
BMS21	3.50	N.D.	N.D.	Long	Control cut

BMS14

As a first attempt at reducing the delay period the standard pattern for cut, helper and diamond holes was fired using short delay No.'s 1 - 7 to replace the long period delay numbers normally used.

Although the cut holes pulled to their full depth the helpers and diamond holes left a series of benches.

It was concluded that a 25 ms interval between cut-helper and helper-diamond holes was inadequate to allow the broken rock to clear from the excavation.

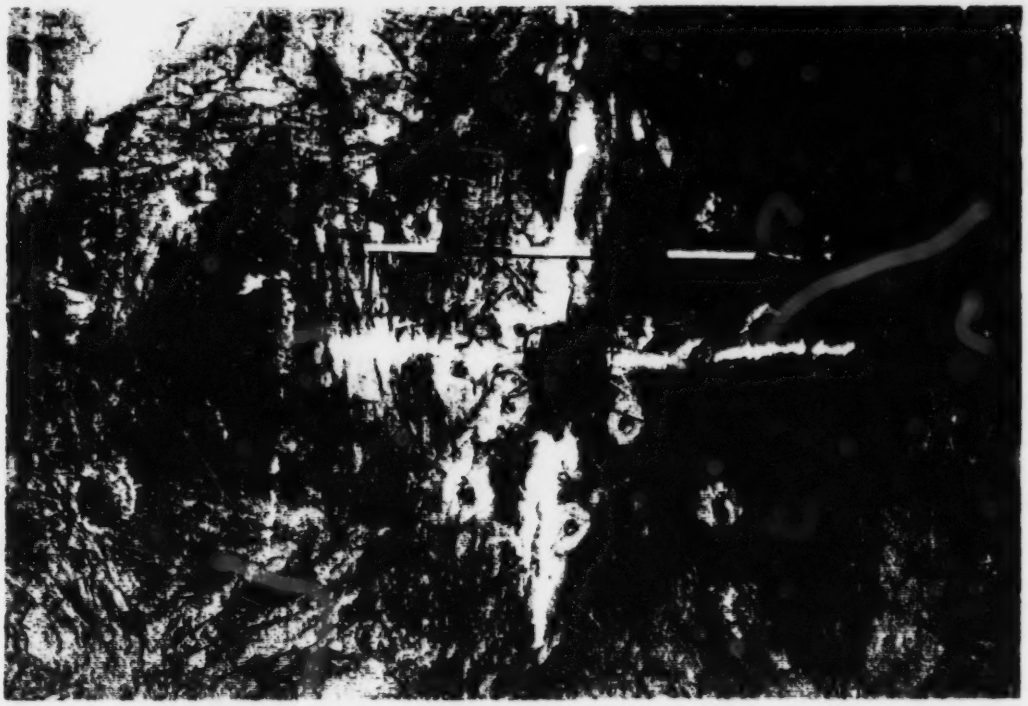
BMS15

This experiment was used as a control. The standard pattern and timing for the first twelve holes (cut, helpers and diamonds) was fired as per a typical development round.

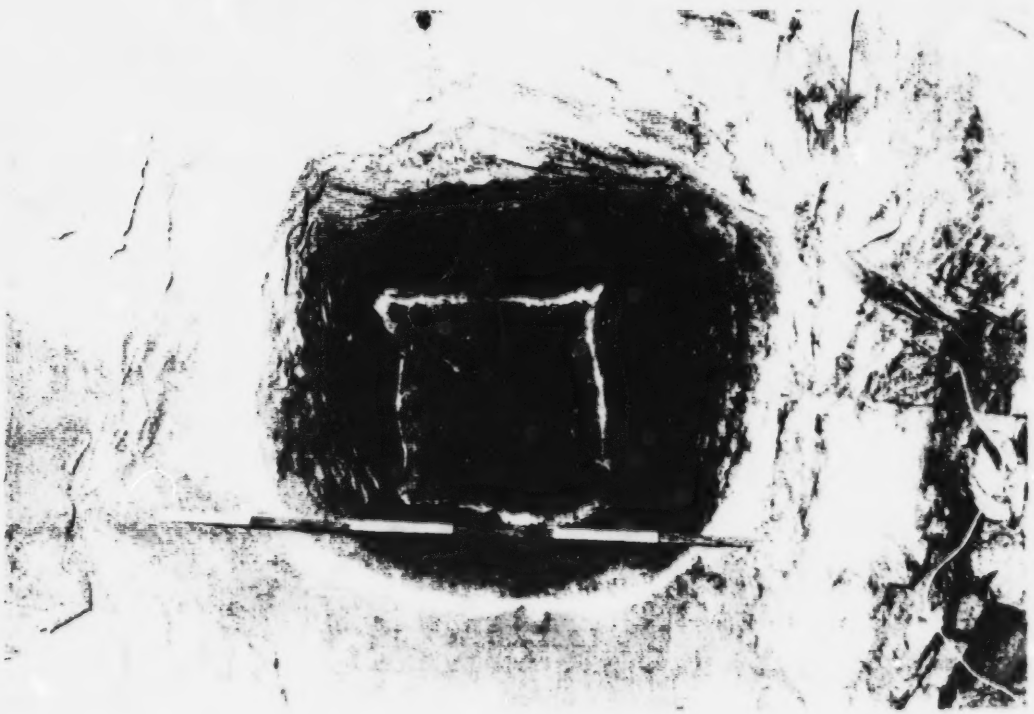
Although the two No. 5 delayed holes failed to detonate the shot pulled to full depth, leaving no bootlegs from the cut or helpers with bootlegs 75 mm - 175 mm from the diamond holes.

As discussed in Section 3.0, this confirms the close helpers are only needed as insurance. Where all cut holes perform correctly, the close helpers can misfire without causing any problems with the remainder of the round.

The control shot pulled to full depth with regular walls and very short bootlegs. The design and the results are shown on Photographs 2 and 3. Experimental shots must therefore aim at repeating this performance.



Photograph 2 Control Cut Design, Long Period Delays
Standard Design at BMS (R. Jessulat)



Photograph 3 Experimental Result of Control Cut Design

BMS16

This experiment was designed to allow delays of 100 ms between groups of holes. Unfortunately due to an error in quoted delay times only 25 ms was allowed between the helpers and diamond holes.

Nevertheless, benches remaining in the excavation were smaller than those left in BMS14 and as such the shot was a significant improvement. The improvement may however have been due in part to the cross grain direction of the shot, other trials being along grain. (BMS personnel indicate that, in their experience, developing across grain is easier than along grain, though results obtained during Stage 1 do not support this, as identical rounds were drilled for both situations).

BMS17

This experiment was of the corkscrew design and represented the only radical departure from standard practice trialled. The hole pattern as drilled is shown in Figure 20 - the design was taken from the El Soldado Mine in Chile where the design has become a standard for that mine. Previous tests indicated that the standard pattern was not entirely suitable for short period delays. Increased time intervals between holes appeared necessary to ensure breakage to full depth. The corkscrew cut however has the potential to be fired in 250 ms - 300 ms, based on the results from the El Teniente Mine in Chili.

As listed in Table 13 the cut only pulled to approximately half of the design depth. The mechanism by which the corkscrew cut operates is not entirely understood and hence modifications required to improve the design are not obvious. However, increased charge hole diameters are thought to be a logical starting point.

The results of the corkscrew were disappointing and hence attention was returned to the standard drill pattern. Photographs 4 and 5 show the design and results of this experiment.

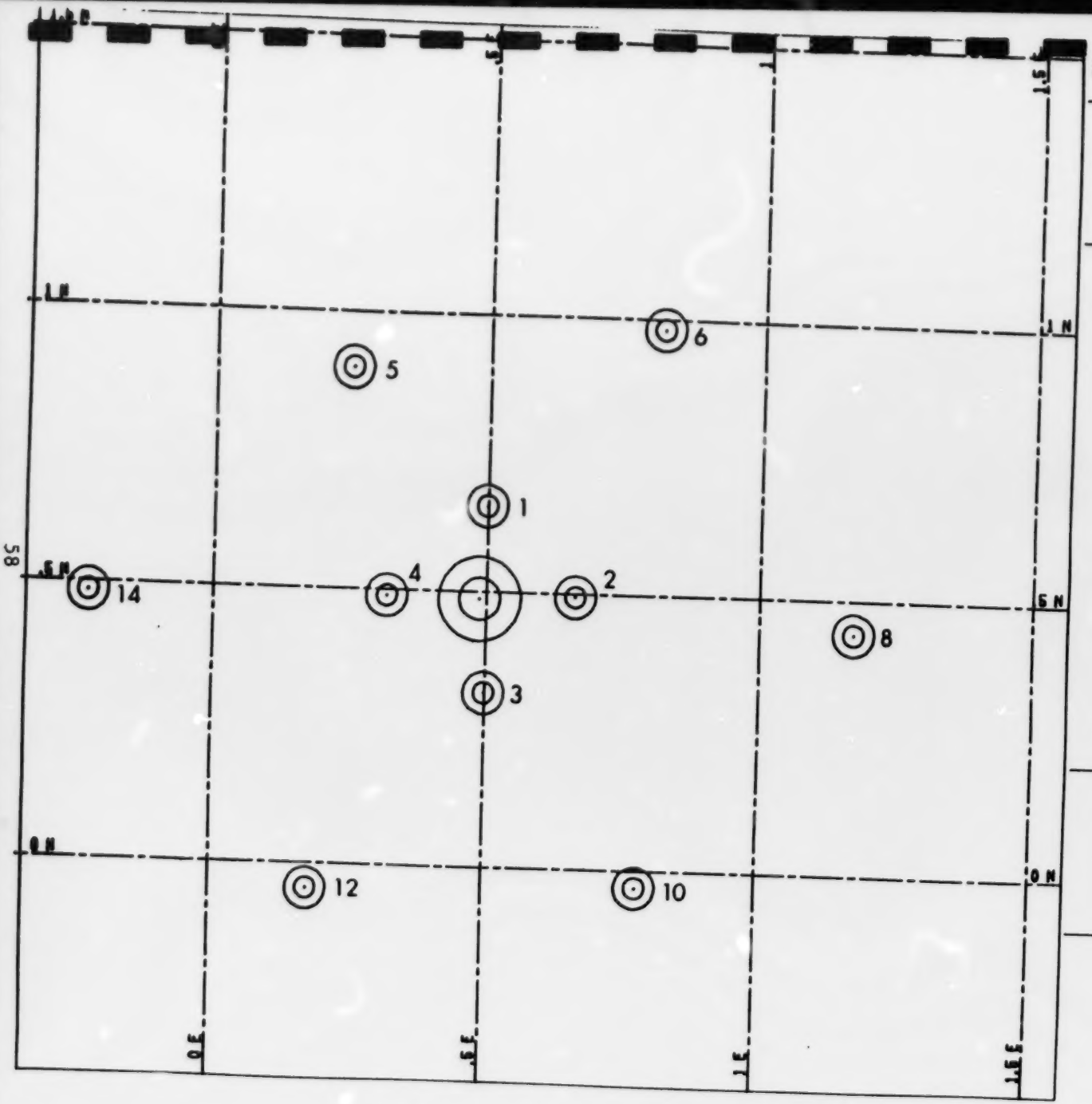
BMS18

This trial was designed with 100 ms delays between groups of holes. In addition the two helpers normally close to the cut were moved out towards the bow formed by the diamond holes.

The modified pattern was an attempt at avoiding misfires and low order detonations (LOD) in the close helpers.

No vibration records are available for this shot (equipment malfunction). It is however suspected that the No. 2 and No. 4 holes misfired or at least suffered LOD's as traces of blasting agent were found in these holes. The helpers and diamond holes left significant benches in the excavation - refer Table 13.

It was concluded that due to partial failure of the No. 2 and No. 4 holes excessive burdens remained in the cut. Normally, close helpers remove remnant material but in this design the helpers were too far from the cut. Again this result emphasizes the need for close helpers as insurance.



Scale: 1:10

16-Nov-1989

BMS#17 : 21/09/89

Design: 249-8 Stope North

Plot of: Holes

Prepared by: JKMRC

Figure 20
Hole Layout as Drilled:
Corkscew Design

KEY TO ANNOTATION

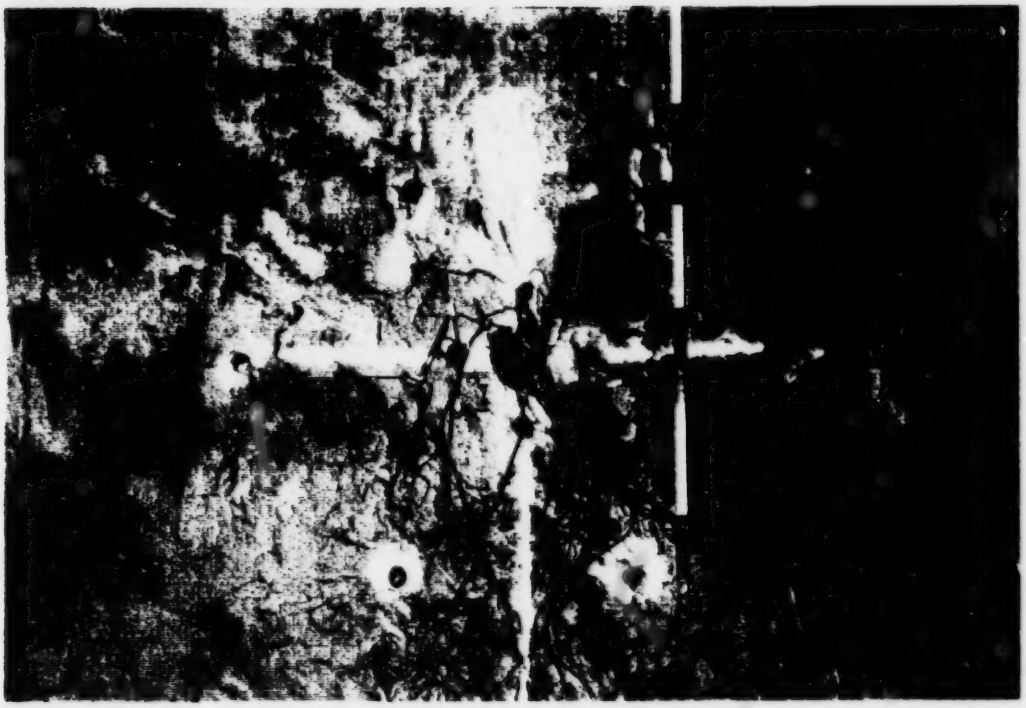
Delay Number

BMS#17.GEN
BMS#17.FEA
BMS#17.HOL
BMS#17.EXP
BMS#17.TIE

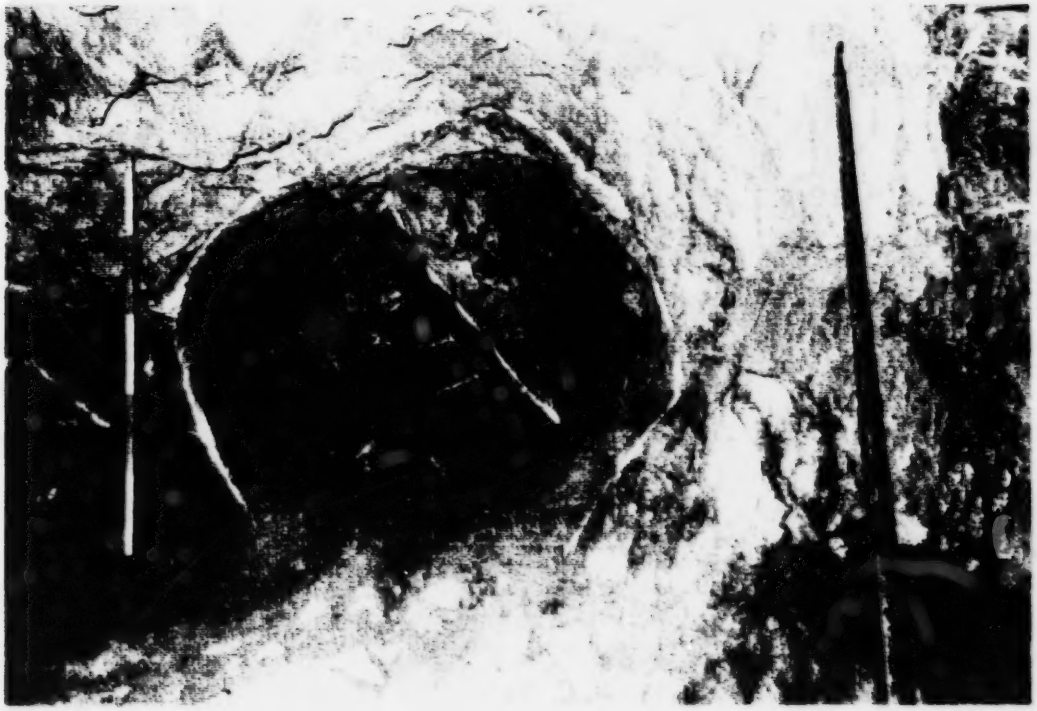
JKMRC

3x3 V1.2

Ph. 61-7-870-0888 Fax. 61-7-870-5775



Photograph 4 JKMRC Corkscrew Design Trialled at BMS



Photograph 5 Experimental Result of JKMRC Corkscrew Design

BMS20

To address the problem of previous experiments where misfires and/or poor detonations were occurring in the central four cut holes, attention was directed at these holes in isolation.

In this event three separate cuts were fired each consisting of four charge and four relief holes. Unfortunately no vibration traces are available and hence correct hole detonations cannot be confirmed.

The first cut (BMS20-1) was fired on a single delay in the hope that simultaneous firing would stop interference between holes. As listed in Table 13 the 0.65 m advance eliminates this design from further consideration.

For the second cut (BMS20-2) relief holes were positioned between the charge holes in an attempt to shield them from interference. Again only a small advance was achieved.

The final cut (BMS20-3) was drilled with two close charge holes and two slightly further away from the relief holes. As with the others only a small amount of ground was pulled.

Following the results of these cuts it was concluded that the ground at BMS was unsuitable for very fast breakage. Future designs would therefore require additional clearance time between holes.

Testing of short period delays was terminated at this point as a significant sulphide dust ignition was initiated by this experiment. Section 3 further discusses the implications of the dust explosion.

BMS21

This test was fired as a control. It consisted of four charge and four relief holes in the standard pattern. The cut experienced no misfires or deflagrations and pulled to full depth.

Any possibility of misinterpretation of BMS20 results are thus eliminated. Further, this result confirms the cuts failed due to insufficient timing between holes.

2.3.4 Summary

The results of the experiments of Stage 2 have shown that the rock at BMS on 2-sub is not suitable for fast breakage. The rock requires plenty of time for relief. The rock is also highly attenuating which means that most of the shock energy is absorbed in the matrix. (These conclusions could not be reached at the beginning of the Project based on the geology, since there has been almost no work done on the relation between blast performances and geology for cut blasting).

Trials with the short period delays were terminated due to sulphide dust ignition. If they had continued it is estimated that an excavation out to the diamond holes would require approximately 500 ms. Past this point it may have been possible to decrease the time between holes as outer holes are only slashing into the excavation.

In terms of a full development round it is thought possible to reduce the overall time down to between 2 and 3 seconds as opposed to the current 6 seconds. Further reductions in timing may be possible if the corkscrew idea is continued but considerable effort may be required in adapting this technology for Brunswick.

With the few trials that were completed, short period delays were noted to increase fines generated in the cuts and also expelled material further out of the excavation.

A higher fines concentration increases the risk of a dust ignition and with increased fly rock the risk of damage to services (ventilation tubing and pipes) is also increased, both negative factors for short period delays.

Even ignoring the sulphide dust problems, the short period delays do not offer any advantages from the rock breakage perspective at Brunswick. Therefore, without any other external incentives to pursue reduced timing, it is recommended that BMS continue with current practice using long period delays.

2.4 ROUND DESIGN AND PRODUCTION TESTS (STAGE 3)

2.4.1 General

The main objective guiding the experiments during Stage 3 was the optimization of the cut in order to eliminate misfires.

This stage was production oriented, and its purpose was to try to reduce the drilling time. No attempt was made to reduce secondary dust explosions.

The idea was to verify a new cut designed by Khilström([3]) which used only one relief hole. In addition, a decoupled stick explosive (Trimrite) was tested, and was aimed at reducing the powder factor in the cut and increasing efficiency.

2.4.2 Experiments Description

A total of 3 cuts and 6 rounds were blasted during this test.

Figure 17 illustrates the location of the cut tests (BMS 23, 24 and 25). The tests included two double spiral cuts and a corkscrew design with long delays. The last double spiral cut was shot with Trimrite.

The double spiral cut was then moved to production where 6 full rounds were drilled and blasted. Four of these cuts were loaded with Trimrite.

2.4.3 Results

BMS 23, 24 and 25

According to Khilström, the double spiral cut is the most efficient cylinder cut, allowing each hole to blast towards the previously blasted hole. The first hole is blasted towards the single relief hole.

A modified double spiral cut was drilled for Test BMS 23. The modified cut contained two relief holes instead of a single one as an extra precaution. The holes were loaded with Trimrite 32 mm cartridges (a watergel explosive used for back control and manufactured by ETI). This decoupling of the charge had the effect of reducing the energy of the explosion.

The double spiral cut pulled well all the way to the back. All the holes were detonated properly.

In Test BMS 24, a second modified double spiral cut was drilled but this time loaded with a smaller diameter Trimrite cartridge (25 mm). This provided increased decoupling and therefore a further reduction in explosive energy.

The cut did not blast well and the pull was less than 2 feet.

In the last cut test, it was decided to retry the corkscrew design with long delays and the Trimrite explosive.

The cut pulled well, but the fragmentation was poor and a slab of intact rock remained at the face (possible from the last cartridge being ejected from an adjacent hole).

Production Tests

With the experience gained in the previous tests, it was decided to move to production for testing.

The double spiral with 32 mm Trimrite was chosen for extensive tests. The 25 mm Trimrite was discarded because of its insufficient strength and the corkscrew was also omitted because of its unreliability.

Several rounds were drilled and blasted in various locations on the north side of 2 Sub. The rounds were blasted in both ore and waste sulphide (along and against the grain). Both the original double spiral cut (one relief hole) and the modified double spiral cut were blasted successfully with Trimrite (only the cut was loaded with Trimrite, the slash holes were loaded with Anfo).

During the tests, the drilling time was recorded, to evaluate the production benefits of the new cut. The time was reduced by approximately 20%. However, as the cut requires increased drilling precision, Ron Jessulat estimated that the increased productivity was offset by the risk of cross-drilling in the cut. This is of considerable concern with the old pneumatic Jumbos being used. The mining crew therefore decided not to adopt the new cut.

Precision of drilling becomes more and more important when the number of relief holes is reduced to one. This is even more significant in an absorbing rock matrix where a burn cut cannot be used. If the first hole is not drilled perfectly parallel to the relief hole, the entire cut can freeze up.

During this stage, both Trimrite and Anfo were used. Vibration monitoring showed that it was possible to eliminate misfires in the cut by reducing the amount of energy used (by using 32 mm Trimrite). Similar tests with the normal round used at the mine also demonstrated that it was possible to eliminate the misfires of the helpers by using Trimrite. It would therefore be possible to reduce the number of helpers drilled if Trimrite were used. However, the added cost of the explosive and increased time of loading reduces the benefits of this smaller cut.

Some Lomex was also tested. Lomex is an Anfo/polystyrene bead mix, commonly used for back control, which reduces the energy of the Anfo. There was one area on the 2-Sub N where a dike intersected the waste sulphide contact and drifting inside this rock created severe stability problems.

The mining crew tested various perimeter hole configurations with Lomex and succeeded in controlling the back with a series of 10 holes drilled at the back (instead of the usual 6) followed by a row of 5 holes drilled 1 meter below it.

2.4.4 Summary

- It is possible to eliminate the number of misfires in the cut by lowering the energy factor.
- Lomex can be used successfully to control the back in difficult rock conditions (the key is to increase the number of perimeter holes and increase the distance of the breast holes).
- Both a corkscrew and a double spiral cut can be drilled in BMS-M's geology as long as long delays are used.

- The advantage of a cut with fewer reamer holes is offset by the mechanical limitations of the pneumatic 3-boom jumbo, the increased precision required and increased loading time.

The work provided some important modifications which could have significant application in a less absorbent matrix. The exact geological make-up of this matrix and the properties which make it absorbent are not well known. Intensive testing at various mining sites would be required to develop empirical formulas which better describe the rock behaviour in development blasting. Empirical curves similar to the Livingston cratering curves could be developed, and information could be obtained from studying the cratering tests that have been performed at various mine sites around the world.

DISCUSSION OF RESULTS

3.1 SECONDARY SULPHIDE DUST IGNITIONS

Throughout the project, several round table discussions between members of the field group and respective supervisors from NTC and JKMRC were held to review results and reset their search objectives. The following discussion was taken from these meetings.

A) **Frequency of Incorrect Detonations**
The initial aim of the project was to identify any misfires and/or deflagrations which were possibly contributing to dust ignitions. Stage 1 demonstrated that only a small percentage (6%) of holes were detonating incorrectly.

B) **Prospect of Reducing Misfires**
The random occurrence of the non- or poorly detonating holes suggested it would be difficult to reduce their frequency. Further it is considered an unrealistic goal to totally eliminate such occurrences. Therefore, if these holes are responsible for dust ignitions, improved blasting techniques will not entirely eliminate the problem.

C) **Relevance of Geology to Risk**
The crew monitored during Stage 1 incurred approximately four relatively minor dust ignitions. In other locations however the same crew have experienced dust ignitions following every blast, while following the same practices. This clearly indicates the role of geology as a major contributing factor.

D) **Time of Occurrence of Secondary Explosions**
One secondary dust ignition was captured on the video unit during Stage 1. The ignition occurred at approximately 2900 ms. The vibration results recorded the expected four No. 8 delays (nominal time 2650 ms) between 2677 ms and 2761 ms. Four No. 9 delays (nominal time 3050 ms) were also recorded and detonated between 3075 ms and 3176 ms.

The sulphide dust ignition therefore appeared to start at a time when no blast holes were detonating. Further, the holes on No. 8 and No. 9 delays all recorded good vibration peaks indicating proper detonations.

It should be mentioned, however, that in pyrite ignition tests (for 20-L chamber) reported by Dr. K. Mintz of CANMET, the minimum induction time calculated was about 30 ms. It required an additional 70-600 ms to reach the maximum pressure, depending on the concentration and particle size of the dust. Therefore, for the sulphide dust filmed at BMS-M, it is possible that a delay of more than 200 ms went by between the time of ignition of the dust by the detonating hole and the appearance of the flame from the dust explosion. One No. 7 delay (nominal time 2275 ms) did misfire, and if Dr. Mintz's theory is accepted, 600ms could have elapsed between the dust ignition and the appearance of the flame. The time lapse noted above is valid for the full scale case, it may have been the source of ignition. It is therefore not certain, based on this video information, precisely when the dust ignition occurred.

E) **Reignition of Blast**

The dust ignition filmed dies out prior to all holes detonating on the face; the top right hand and left hand holes (delay No. 14) detonated after the dust explosion appeared to die out.

The dust explosion then either returns to the face or a second ignition occurs at 6040 ms, some 558 ms after the last hole detonated. The source of this second flash is at present unknown.

F) **Effect of Reducing Delays**

The series of tests to reduce the timing of the round was undertaken because it was considered that limiting the travel time of the dust produced during a blast would confine the ignition to the face area and hence minimize it. Video recordings during these experiments however showed dust to be expelled rapidly from the face and to travel a considerable distance within 200 ms.

A dust ignition occurred during Experiment BMS20. This trial consisted of three cuts separated by short period delays. The last hole to detonate had a nominal time of 275 ms. The dust ignition therefore occurred within the 275 ms or shortly thereafter due to the final hole.

Trials to blast the round within 500ms proved unsuccessful. It was therefore concluded that reducing the timing to the 200 ms required was not feasible.

G) **Prospect of Reducing Dust Generation or Sulphide Levels**

A development round is driven to access an orebody and hence there is generally little opportunity to re-direct a drift to avoid passing through reactive material.

Dust is produced around all blast holes and hence is unavoidable. The concentration of the dust may be reduced however if the cut was fired separately to the main part of the shot. This would seriously interfere with production and to date has not been pursed as a possibility. Avoidance of generation of an explosive dust cloud by modification of blast design appears unachievable.

H) **Elimination or Reduction of Oxygen**

Some suggestions of filling the drift with inert gases or foams etc..., have been presented but again these systems would seriously interfere with the production cycle.

I) **Very Long Delay to Permit Dust to Settle**

Very long period delays with 20 or 30 seconds between detonations have been investigated by NTC and found to be impractical given current technology.

J) **Benefits of Inert (Gelled Water) Stemming**

Review of video footage highlighted the use of inert water-gel stemming for reducing the flame emitted from a hole. If correctly tamped into the hole the flame was reduced to a small flash as opposed to a large yellow flame drifting up into the dust cloud.

The Stage 3 segment of the project presented a limited opportunity for further testing. The final blast monitored (BMS29) was correctly stemmed with tamped water-gel cartridges. A small sulphide dust ignition did however occur. This result indicates that other material expelled from the hole (as well as, or instead of the flame itself) can cause the ignition.

K) **Benefits of Water Sprayer**

In parallel to the work on blasting techniques, the efficiency of the water sprays was evaluated for two crews on the 1000 mL no. 2 sub as part of this project.

The water spray with the timing device was designed by Don Gagnon of BMS-M. The spray is triggered electronically off the B-line and directed at the cut. Figure 21 illustrates the spray system set-up in a drift.

Throughout the 12 weeks of field work, our research team along with Ron Jessulat's crew tested the spray in medium to very high sulphur zones. Three dust ignitions occurred during this time. Failure of the water spray was partly attributed to incorrect usage, as for each of these events, the spray had not been set-up properly.

Viedo footage showed that with the nozzle secured on the floor, the water jet does not totally penetrate the dust cloud. Attaching the nozzle to the back close to the face ensures the water jet is cooling material as it leaves the holes.

Verbal communication with supervisors and miners at BMS suggests the water spray has been effective in certain areas where other passive methods have failed. As an extension to the field program at Brunswick, the use of the water spray was evaluated by Don Gagnon and Ron Jessulat's mining crew for a two month period.

The following points summarize the result of this evaluation period:

(i) The water spray, if used correctly, is efficient at eliminating sulphur blasts in medium to high risk zones (25-35%). In very high risk zones containing thick pyrite lenses, the water-spray works approximately 50-70% of the time.

(ii) For very high sulphur zones, a combination of gelled water stemming of the perimeter holes and the water spray appears to eliminate completely the sulphur blast. This must be confirmed over a longer test period.

(iii) Proper installation of the water spray with respect to distance from face, location of the hose and direction of the spray is critical to its success during a blast.

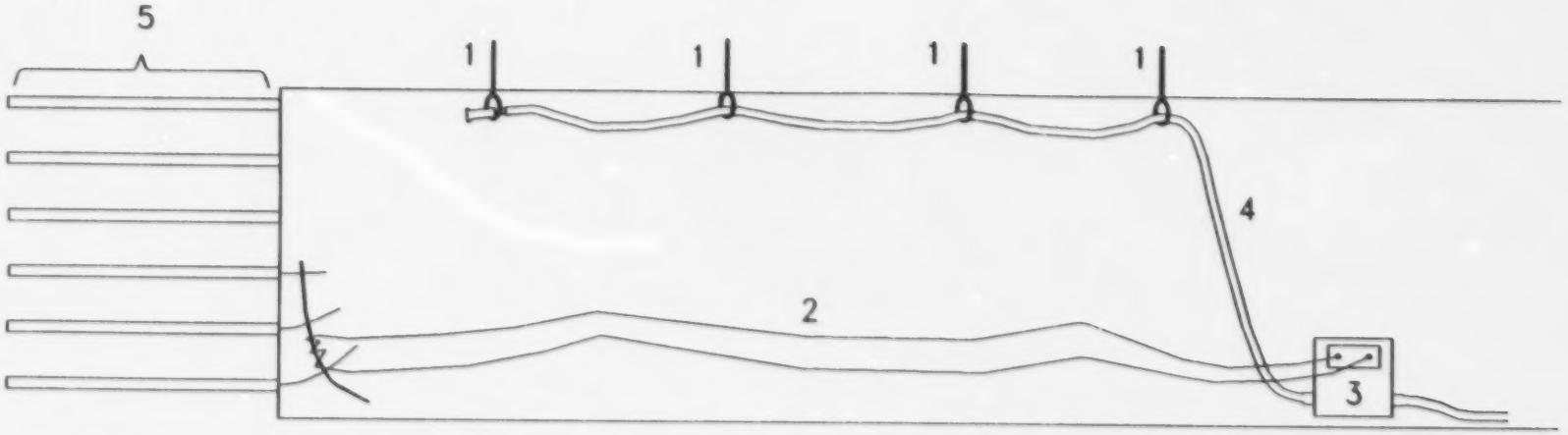
A different crew tried out the spray on the no. 2 sub (1000 mL) and had limited success. According to Mr. Jessulat, the problems were due to improper setting of the spray.

Further work to optimize the installation is on-going at Brunswick. Mr. Gagnon is also looking at the possibility of using a 1 1/2" diameter solenoid valve instead of the present 1" valve in order to obtain a greater volume of water ejected at the face.

L) **Benefits of Low Temperature Explosives**

In addition to the water spray, the use of an explosive with reduced detonation temperature may also help reduce sulphur blasts. Laboratory testing has shown slurries and emulsion explosives to have such a lower detonation temperature that reactive dust clouds are not ignited.

A few rounds using Trimrite (packaged slurry) explosive were tried in Stage 3 and no dust ignition were recorded. Such limited tests are not conclusive but are encouraging.



1. EYE BOLT
2. BREAK WIRE
3. WATER SPRAY TIMER
4. WATER HOSE
5. ROUND

Figure 21 Spray System Set-up

3.2 FUTURE DIRECTION

The discussion presented in Section 3.1 highlights several areas where current knowledge is incomplete, i.e. the exact source of ignition, and the relationship between an ignition and the damage caused. However, the following recommendations will, in the opinion of the authors, help to eliminate the risk of secondary sulphide dust explosions.

1 - The high pressure water spray, as designed by Don Gagnon, has good potential for being successful. In very high sulphide zone it should be combined with gelled water stemming in the perimeter holes.

2 - In high risk situations where ineffective is found to be explosives with a lower temperature of detonation, (slurry or emulsion) should be evaluated.

The use of lower temperature explosives is supported by laboratory experiments and had been suggested by several experts both before and after the project. Any trial of a new explosive type would need to be conducted over an extended period to ensure several blasts in very high risk ground were monitored.

3 - To assist in better identification of high risk areas, a geological map with an overlay showing locations of secondary sulphide dust ignitions would be useful. The optimum solution will depend on the degree of risk in different sections of the mine, and such predictive information is viewed as essential.

REFERENCES

1. Saindon, J.P., Noranda Technology Centre. Continuous Analysis of Gases, Dust and Flame During and After Blasting in Canadian Sulfide Ore Deposits. DSS Project No. 14SQ.23440-7-9140, October, 1989.

2. Singh, Wondrad. Review of Drift Blasting Practices in Canadian Underground Mines. Laurentian University, School of Engineering, Sudbury, Ontario, 1989.

3. Langefors, U., Kihlström, B. Rock Blasting. John Wiley & Sons, Inc. Third Edition, 1978.

4. Weiss, E.S., Sapko, M.J., and Watson, R.W. Blasting Agent for Use in Flammable Atmospheres, USBM, Pittsburgh, PA, 1988.

5. JKMRC. Evaluation of Development Rounds at Renison Mine (Renison Goldfield Consolidated), AMIRA Report, Melbourne, Australia, 1987.

6. JKMRC. A Preliminary Study to Optimize Blasthole Distribution in the Development Rounds at the Renison Mine. AMIRA Report, Melbourne, Australia, 1988.

7. JKMRC. Development Blasting Research at the Andina Mine. AMIRA Report, Melbourne, Australia, 1987.

8. E.I. du Pont de Nemours & Co., Wilmington, DE, 19898. Blasters' Handbook, Distributed by ABA Publishing Company, 1980.

9. Bullock, R.L. Fundamental Research on Burn Cut Drift Round. Third Annual Symposium on Mining Research. Bull. University of Missouri, School of Mines and Metallurgy, 95, p. 84-121, 1958.

10. Holmberg, R. Underground Mining Methods Handbook. Society of Mining Engineers, Section 7.5, Chap. 1, pp. 1580-89, New York,. 1982.

11. USBM. Underground Blast Design. Jour. of Explosives Engineering, Vol. 2(5), January - February 1987, p. 14-17.

12. Keller, R.L. Are You as Efficient as You Can Be ? Dynatech Corp., Haverstraw, N.Y., 1988.

13. Hoek, Brown. Underground Excavations in Rock, The Institution of Mining and Metallurgy, London, 1980.

14. Favreau, R.F. "Blastpa, un système pratique d'optimisation en sautage: étude de cas". Troisième session d'étude sur les techniques de sautage, 30-31 octobre '80, Université Laval, Québec.

15. Andrieux, P. Résumé sommaire sur le simulateur de sautages Blastpa. Internal Report, Centre de Technologie Noranda, janvier 1989.

16. Atlas Powder Company, Explosives and Rock Blasting, Copyright 1987, Dallas, Texas, U.S.A.

17. Private Communication between Jean-Paul Saindon and Gord Jorgenson, Manager, Blasting Physics Explosives, CIL, August 1989.

APPENDIX A

MEMO TO PROJECT MANAGEMENT
ON
FINAL PROJECT STATUS

3 NOVEMBER 1989

REDUCTION OF FREQUENCY OF SULPHIDE DUST EXPLOSION BY IMPROVED BLASTING TECHNIQUES

Memorandum to : George Ouelette and
Project management
From : Gideon Chitombo (JKMRC)
Yvan Dionne (Noranda Tech. Center)
Subject : Final Project status
Date : 3rd Nov 1989

The above project officially ended 27/10/89.

The main project conclusion was that water sprays using the blast spray timer developed by D. Gagnon of Brunswick mining are at present the only and most effective means to reduce the occurrence of dust explosions.

A limited test period is therefore required to fully test the critical variables associated with the effective use of the sprays and to develop a practical, economic and quick means to install sprays on site.

The critical variables identified include:

1) The amount of water in liters per minute (or gpm) required,
2) The distance between the spray nozzle and the face,
3) The size and location of the spray area.

We feel the above variables need investigations before the spray system can be strictly enforced and transferred to all sections of the mine.

The test proposed extends until the 14th of November 1989. Yvan of noranda will assist with the investigations.

To ensure that the above goals are pursued in a proper manner we recommend that Ron Jessulat's crew continue with the currently agreed average bonus until the 14 th of November 1989.

After this period the proven system should be tested over a wide range of sulphur dust explosion prone areas. During this time, it would be the mine's responsibility to enforce this long term study and to ensure proper documentation before the use of sprays can be strictly enforced and transferred to all sections of the mine. Efforts should be made to adopt the proven system to stope blasting.

With thic momo is a more detailed project summary for your information. Final reports will be issued in due course.

c.c. NTC
JKMRC
Brent Kristof
Fred Hermann
Bob Fisher
Glen Crowther
Don Gagnon
Ron Jessulat
J.P. Basque

Gideon P. Chitombo

November 3rd 1989

REDUCTION OF FREQUENCY OF SULPHIDE DUST EXPLOSION BY IMPROVED BLASTING TECHNIQUES

A joint Noranda T.C. / JKMRC / BMS project.

PRELIMINARY PROJECT REPORT

1 - INTRODUCTION

The project aim was to reduce the frequency of sulphide dust explosions by improved blasting techniques. The JKMRC was subcontracted by Noranda Technology Center's mining and environment group. The JKMRC's involvement was officially over a 12 week period. In practice, however the overall project was carried out as a strict joint effort between JKMRC, NTC and BMS. The designated Brunswick mines development crew was headed by Ron Jesulat. The crew members were Frank Godin and Donald Plourde. Cletus Morrison assisted during the initial stages of the project.

2 - PROJECT AIM

The monitoring work and subsequent experiments were designed around two basics assumption:

i) Sulphur dust explosion are more likely a result of poor blasting practices in particular blastholes which either misfired or deflagrated (low order detonation).

ii) The timing or speed of extraction of a development round was also thought to be a critical factor. Extracting a round, in particular the cut, at rapid rates would limit the travel of the reactive dust produced and thus the extent of any dust explosion.

These basic assumptions were based on Noranda's past experience or observations and for all practical purposes appeared logical and worthy of pursuit and investigation.

The field work was carried out in three stages:

i) Stage 1: involved general monitoring and assessment of the blasting practices of the designated Brunswick development crew.

ii) Stage 2: involved implementing short period delays (25 ms) supplied by ETI.

iii) Stage 3: involved experimenting with new cut designs in particular the so called cork screw burn cut which utilizes short period delays.

3 - PROJECT RESULTS

The details of work conducted including individual results will be discussed in the final reports currently being prepared.

The instrumentation package used during the project included:

a) Vibration monitors, with geophones as the primary sensors, and,

b) A video camera system put together by Noranda Technology.

The vibration monitoring results strongly suggested that very little could be done, with respect to blast design optimization, to prevent the occurrences of dust explosions.

A very low percentage misfires and deflagration was recorded from the monitored blasts. All together 497 holes were monitored during stage 1 representing 13 successfully monitored blasts. Of those whey were 26 recorded misfires and 6 deflagrations representing 5 % and 1 % misfired holes and deflagrated holes respectively.

Most of the reactive dust was generated by the cut. There was documented evidence, through video and vibrations records, that the explosions occurred or were triggered by holes just outside the cut area. From this it would appear that there are no positive advantages in speeding up the extraction or the full round itself.

During these timing trials, it was observed that cuts initiated with short delays produced even finer and therefore more reactive dust. Further more, the dust was thrown further into the drives, a recipe for explosions covering a much bigger area.

From a practical or operational point of view, cut experiments using short periods delays did not pull the ground to full depth. This included the cork screw and some of the proven cut designs used by the Brunswick crew. This is more a function of the brunswick rock type than the cut design per-se. True burn cuts appear not to work in this rock. The rock seems to require relief and relatively long delay time to successfully pull the cut.

The cut area itself is traditionally designed with very high power factors to allow for the crushing and ejection of crushed rock to give the initial expansion volume for the blasted rock. Unfortunately in the case of Brunswick ore the finer the crushed particles are, the more reactive the resulting dust is and therefore more susceptible to explosion. A relatively lover strength explosive, trimrite 2 supplied by ETI, was tested in the cut in an attempt to reduce the cut power factor and hence the creation of these "fine" reactive dust particles. The test however proved unsuccessful. Similar amount of dust was created.

4 - TENTATIVE CONCLUSIONS

i) A very lov percentage of misfires and deflagrated holes was recorded during the project. This is in particular reference to Ron Jessulat's crew. Although fever blasts were monitored from other crews the percentage of misfires and malfunctioning holes was still small.

ii) In spite of the near perfect performance of the blasts monitored, dust explosions still occurred.

iii) The above results suggest that little can be done in terms of blast performance optimization to prevent the occurrence of dust explosions. The occurrence of dust explosions cannot be directly or at least easily related to blast performance. As a general rule however, good blasting practices should continue to be enforced.

iv) There appears also to be no direct link between the timing of the blast and the occurrences of the dust explosions. The use of short period delays, in this case 25 ms ETI delays, cannot be justified sorely for the elimination of sulphide dust explosions in Brunswick mines.

v) From an operational point of view the Brunswick rock, at least on 1000 M level, cannot be efficiently broken and pulled with fast delay times. This is in reference to cuts. Different and even proven cuts suffered the same effects.

vi) The documented blasts showed that the explosions are triggered by holes immediately outside the cut areas.

vii) The most effective means to reduce the frequency of sulphide dust explosions appears to be by passive means.

viii) Of the passive means experimented, water gel stemming and water sprays, the high pressure sprays seem to be most effective.

ix) The mounting of the high pressure spray also appears to be critical. From the limited tests conducted roof mounting is more effective.

x) The sprays are more effective when aimed directly on the cut area.

xi) Other critical variable, to be investigated, regarding the spray include:

a) The distance between the spray nozzle and the face,
b) The size of the area sprayed,
c) The amount of water in Liters/min. (or GPM) required.

5 - FUTURE TRIALS

a) Investigate the most practical, economic and quick means to roof mount the pressure spray hose.

b) Investigate some of the identified critical variables with water sprays:

c) The sprays should be tested in a wide range of sulphur dust prone area, e.g:

1) High risk : uneconomic grade < 8 % Pb + Zn,
2) Medium high risk : low grades area 8 - 10 % Pb + Zn,
3) Medium low risk : Medium grade, 10 - 14 % Pb + Zn,
4) Low risk : High grade, > 14 % Pb + Zn.

d) A large (documented) data base is required in order to determine fully the effectiveness of the water sprays in all geological or explosion risk areas, before the technology can be strictly enforced and transferred to all sections of the mine.

e) The option to use lower heat generation explosives as a means to reduce the occurrence of dust explosions should be kept open.

f) The proven spray technology should be adopted to stope blasts.

6 - PROPOSED FOLLOW UP WORK: IN-HOUSE WATER SPRAY TESTS

a) Investigate all variables in some controlled and logical manner.

b) Document results, paper and video (Brunswick system).

c) Develop practical and simples means to establish risk areas.

d) Assign dedicated individual to ensure proper follow-up.

7 - OTHER TRIALS

a) Lomex in cut holes
- to reduce quantity of dust,
- to reduce heat.

Gideon P. Chitombo

G.P. Chitombo

APPENDIX B

DETAILS OF FIELD WORK AT BRUNSWICK MINING

6 AUGUST - 9 NOVEMBER 1989

DETAILED REPORT ON FIELD WORK

The blasts detailed in this report are described in the same order as in the video-tape. References are constantly made to what is displayed on the tape and the reader is therefore strongly suggested to view the tape at the same time as he reads the document.

STAGE 1 (August 16th to September 14th)

Week 1

- First meeting

Work has started on August 17. A meeting with Jean-Pierre Basque of Brunswick Mines was held on the 17th. A general project overview was provided.

Minutes of the meeting:

Stage 1 : General blast auditing.

(i) To observe and record the current heading blasting methods and practices:
- Blast geometry (pattern)
- Number of holes
- Types of cuts
- Hole size (diameter and length)
- Explosive types
- Initiation system
- Delays
- Loading practices
- Dust control methods
- Advance or pull
- Wall condition (damage)

(ii) To monitor the performance and effectiveness of the current methods and practices through:
* Vibration monitoring
  - Vibration amplitude level
  - Blast hole sequence of detonation
  - Misfire detection
* Video filming
  - Blast performance
  - Flame detection
  - Dust occurrence
  - Effectiveness of dust / flame control methods
* Post blast survey
  - Advance / butt
  - Fragmentation / wall condition

(iii) To analyze the current design and delays through the

JKMRC software (3x3d).

(iv) To modify current designs (only if required) based on the monitoring results and discussion with the drilling and blasting crew, engineers and production.

(v) To monitor the recommended changes.

(vi) To establish optimum methods and practices based on all of the above using the current design constraints.

Stage 2

(i) Implement the EXADET ETI detonators to the optimized designs

(ii) Delay scatter or variability tests of the EXADET

(iii) Monitor blast using EXADET:
- Vibrations
- Video filming
- etc. (as in stage 1)

Stage 3

Experiment new designs
- Cuts including corkscrew
- Wall control
- etc.

Note: The cut experiments should be initially carried out in different parts of the mine where failure will not be detrimental to production.

Pat Walker and Mark McKew of JKMRC had to follow an induction course to the mine on August 17 and 18. They were not present during the meetings.

- Second meeting

A second meeting was held on August 18 with George Ouelette. Project goals and requirements were established and the crew and work area were chosen according to project objectives.

Minutes of the second meeting:

The project goals outlined in memo #1 (first meeting) were presented to George Ouelette. George supported the project objectives and the work outlined and scheduled for the three stage of investigation.

Stage 1 (blast auditing) is expected to takes up to 4 weeks. In addition to the outlined goals, George suggested the work include and/or be concentrated on the following:

- Waste sulfide/ore and waste rock development rounds.

- Effect of mining direction (with or against grain).
- Effectiveness of the different dust suppressing methods:
  - Spraying the cut area directly.
  - Curtain of water in the drift.

George Ouelette believes that different cut designs, wall control methods and dust suppression methods have been tried in Brunswick. Different methods have worked successfully in different conditions and not in others. The present project should therefore attempt to quantify the performance/mechanics of the different methods outlined above.

The following test areas have been designated for the field trials over the 10 weeks project. The area are all on 1000m level (3250') # 2 sub.

- 2 sub north drift (waste, along the grain)
- 249-8 and 251-8 stopes (ore, across grain)
- 247 hanging wall access (ore, along grain)
- 1000 fill raise access (waste, across grain)

Reference to a 1000m level plan.

The project team will be working with Ron Jessulat's crew. The crew is composed of:

- Ron Jessulat, drill,
- Frank Godin, load and blast,
- Cletus Morrisson, rockbolter.

A meeting was arranged between the research team, the shift supervisor, Don Gagnon and the crew on August 21st. The meeting was be followed by an underground visit of the test area. George Ouelette will make an arrangement for the miner's bonus (10 weeks or duration of field study).

- General

An arrangement was made with the electrical department to have power stations at convenient locations on the level. These power stations were providing 110 and 220 volts for the video system. Lockers and mining gear were also distributed to ensure that everybody was safely dress for the mine.

Week 2

A meeting with the crew was held on monday August 21st. The crew was then informed of the project goals and what were their responsibilities in the project experiments.

The first part of the week (monday to wednesday) was used to prepare the monitoring setup: installation of geophones, power stations arrangements, carrying the material underground and preparing protective cases for the monitoring equipment. The color camera was tested during the week and failed to function properly. It was replaced by a black and white Pulnix camera. Two blasts were recorded on thursday and friday.

- August 21 blast.

The blast was in waste so the water spray wasn't setup for the blast. The light system has worked properly: it went on as programmed and produced enough light to have a good image quality. The camera had an acceptable image definition and the recording speed was adequate.

The first flash viewed on the screen comes from the B-LINE. The B-LINE is an explosive detonating cord used to initiate all the delays in the round. After the B-LINE is initiated, a considerable amount of dust falls from the back and the walls. This dust blocks the view of the face but the image remains acceptable. The first hole produces a large flash of light which remains 30 ms. The dust ejected from the first hole quickly fills up the drift with thick dust (300 ms). It is not possible to see the second hole because the dust is too thick. No flame from the round penetrates the dust.

- August 22 blast

The initiating set-up was different on this blast because the water spray was used. When the water spray is used, the electric detonator ignites a short length of B-LINE. The short length of B-LINE in turn initiates a NONEL delay (usually the longest delay available) and finally the B-LINE is ignited. The NONEL delay between the two B-LINE provides a few seconds between the initial ignition and the round blast. This time is required to start the water spray and wet the face ahead of the blast. The water spray is turned on by a trigger wire on the short length of B-LINE. When the wire breaks, the water starts to flow for a period of three minutes.

The spray and the lights were programmed to start after the first spark of the B-LINE. As can be seen on the video, the spray hasn't started. The cause of the failure was not found.

On the video the dust which falls from the walls and back can be viewed. A large flame emerges from the first hole and the dust fills the drift within 400 msec. No sound was recorded and the camera was switched off due to vibration after the first few holes. The blast was in waste sulfide.

Week 3

There was several problems with the video system during that week: early system shut-off, poor connections and errors in timing programmation. There was no successfull video-tapes of the blasts. Notheless the vibration monitoring system was OK and three blasts were monitored.

Week 4

- September 5 blast

A video recording was successfull in the 251-8 south access round. The material blasted was waste sulfide. The water spray was setup for the blast. After the first length of B-LINE the water spray can be observed. The hose was on the ground and spraying 3 or 4 feet above the cut area. A small amount of dust falls from the walls and back. The first hole produces a small flash of light. As usual the drift is filled with dust in less than 400msec. The water sprayed forward seems to be mixing with the advancing dust cloud. it probably spreads in the dust quite evenly due to the turbulent atmosphere.

A sulfide dust explosion occurred at 2.9 seconds after the ignition of the blast. The timing indicates that it is around a #9 delay. The #9 delay was right after the cut and helpers. 12 holes were fired when the explosion occur and the volume of blasted rock was approximately 5 m^3 (Approx. 6% of the blast). The explosion came from the middle of the screen and remained for 2.2 second approximately. The explosion is not homogeneous: it looks like patches of fire going back and forth. At the end there is a wave of sparks passing in front of the camera. The sparks seem to go toward the face.

After the sparks, the flame appears to have subsided but it regains energy after one second. It begins at the bottom of the screen and remains for at least two seconds. The camera ceased to operate before the end because of a burnt power cable. There was only one length of vent tubing (100') stained and it didn't require replacement.

- September 6 blast

A video was made but the image and sound were unusable due to a bad contact.

- September 7 blast

The blast was in the hanging wall drift in ore along grain. There were no lights for the blast because the power station wasn't installed yet. The camera was installed far from the face since the video cable wasn't long enough.

With the light from the explosion of the B-LINE, the drift and the vent tubing can be viewed. There is a fairly large spark from the first hole but it is difficult to judge because of the distance from the blast. Once the dust filled the tunnel, no flames propagated through the dust.

- September 8 blast

The blast was in 249-8 north stope and in ore across grain. The camera was placed far from the face to see if a longer recording time was possible. The spray was attached to the back and aimed directly at the cut. The B-line detonation produced a small amount of dust. The first hole produced a small spark which remained for only 30 msec. As usual the dust from the first hole spread to the entire section of the drift.

Week 5

- September 11 blast

One round was taken in the north access. Since it was in the waste, the video was not installed.

- September 12 blast

One round was taken in the hanging wall drift. It was in ore and along grain. There was no light because the power station wasn't connected. Several sparks are visible after the blast.

- September 13 blast

A round is taken in the 249-8 north stope in ore and across grain. The lights did not function due to a tripped breaker on the power station. Some sparks are visible after the blast. The camera was far from the blast. The sound of the blast is covered by the noise of the fans on the soundtrack.

- September 14 blast

The Round is in the 251-8 south stope access in waste sulfide across grain. The water spray operated properly and was aimed 2-3 feet to the right of the cut. The flame of the B-LINE remained longer than usual. Two large sparks are visible at each upper corner of the drift.

STAGE 2

Since the current blasting practices were good, it was decided to proceed with stage 2 of the project. To implement the fast detonating caps an area outside of the production zone, 249-8 north stope north wall, was chosen.

- September 15 blast

A regular cut with helpers was drilled. It was loaded with ANFO and the long delay caps were replaced with the EXADET caps. The material blasted was ore along grain.

The delay betwwen the light system and the cut ignition is too short and the blast is already going when the lights are fully on. The ejection of the rock can clearly be seen. One rock hits the camera and the image is lost for 250 msec. After, the rocks falling on the ground can be seen. The image is fairly clear until the lights turn off (30 seconds) since the vent tubing pushes the air around the camera and keeps the dust away.

The 4 cut holes have pulled to the bottom. The next four reliefs have started to bench and the last four reliefs have benched around 1 meter.

Week 6

- September 20 blast

A control cut was fired earlier in the week. The control cut indicated that the ground was pulled straight through to the bottom with long delays. The material blasted was ore along grain.

The rest of the round was blasted during this test. From the video it can be seen that a large ball of flame was produced by the first hole. The dust does not fill the drift as fast as usual. The amount of dust produced appears to be less significant than usual.

After the blast, a considerable decrease in the amount of dust in the drift could be noticed. It is now clear that the cut is a major contributor to the dust concentration produced by a round. A lower strength explosive was ordered that week to verify whether or not the amount of dust could be reduced by using a less powerful explosive.

- September 21 blast

The JKMRC Corkscrew was drilled and blasted in the 249-8 north stope wall (ore, along grain). The flame of the B-LINE saturates the screen and before the saturation subsides the rock is already being ejected out. The fast caps are around 20 times faster than regular caps and the speed of the camera was barely adequate to capture the blast.

The amount of dust produced by the fast caps is larger than with regular delays. No precise measurements of the amount of dust produced has been done, but inspection of the site after the blast showed that the dust was more finely crushed and had

travelled further. Also there is almost no material left inside the cut, 80% of it is projected out. In a regular cut, over 50% of the crushed rock is left inside the cut. The blast itself was very poor as it did not pull to the bottom. The relief holes contained as much as 1 m of bootlegs. Since this pattern was not successful, it was decided to go back on the regular pattern with fast caps.

- September 22 blast

A standard cut pattern was loaded with fast caps but some delays were skipped to allow the material more time to get out of the cut. The color camera was used to film this cut and since it is less sensitive to light, the saturation problem was partly solved. The first hole produced a large flame approximately 6 meters long. It was estimated from markers at the face that the flame front was moving around 800 km/h. The dust filled the place within 2 seconds.

The cut kicked up a lot of dust again. The cut pulled up to the bottom but the helpers started to bench. From these results it became clear that the rock was not appropriate for fast cap blasting. Notheless it was decided to concentrate on the cut and leave the helpers out to see whether this part could be optimized.

Week 7

- September 24 blast

A test to verify the vibration recording was done. Two holes were compared: one had a 0.45 meter burden and the other was fully confined. The test revealed that the two vibration traces were quite similar and that the burden did not influence maximum pressure of the shock wave.

- September 27 blast

Three cuts were drilled in the stope wall. Each cut had 4 relief and 4 charged holes. The holes were charged with ANFO and detonated with EXADET fast caps. All the holes of the first cut to fire were on the same delay. The second one had 25 ms between each holes. The third cut was modified a bit: two opposed holes were placed a bit further from the center of the cut; each holes were set-up with 25 msec delays. The material was the same: ore, along grain.

None of the cuts pulled more than half of the planned depth. The best performance was achieved with the third cut. A dust explosion occurred when these three cuts were fired. Approximately 1200 feet of vent tubing was lost (burned or ripped) because of the sulfide dust explosion. The extent of the damages extended as far as 600 feet from the blast. The damages

were greater than many sulphide dust explosion from a complete round yet the amount of material blasted was small (≈ 1 m3); about 5 % of a full round.

This test was a major turning point in the project since it showed that the timing could not be reduced sufficiently to prevent or minimize a dust explosion. This concluded our experiments with fast caps and we proceeded to stage 3 where new and more efficient cut designs were evaluated. Although these experiments were aimed more at increasing productivity then at reducing sulphide dust explosions some further work with the sprays and water-gel cartridges was carried out concurrently. A new explosive, TRIMRITE, was also tested. TRIMRITE is usually used for back control: it is produced in continuous tubes or in cartridges.

- Meeting with BMS

A meeting with BMS project management was held on September 29th. The stage one report was presented to Brunswick Mines. Conclusions were as follows:

- All the rounds contained at least one deflagration or misfire (ranging from 1 to 6).
- All the detonators worked correctly and there are few chances that delays overlap.
- The percentage of misfire is very low and it is not considered realistic to aim at eliminating them. An odd one or two will always occur.
- From the video we see that the inert stemming reduces the flame extent and duration.
- One sulphide dust explosion occurred even though the water spray was used.
- The drill, load and blast process was very good and no modification had to be done to the current practices.

STAGE 3

Week 8

- October 3 blast

As a control test, Ron's standard cut was blasted. The flame produced by each hole was clearly visible on the video. The flames were short and remained for a short duration. The cut pulled well down to the bottom.

- October 4 blast

The same cut as October 3rd was drilled but this time it was blasted with TRIMRITE. The flame appears to have the same duration as the standard cut, although of less intensity. Even if the TRIMRITE is less powerful than the ANFO, the cut pulled well. The TRIMRITE was in cartridge and it was loaded manually. The last cartridge was broke to fit perfectly in the hole. The holes were loaded to the collar.

- October 6 blast

A double spiral cut was drilled and loaded. The double spiral cut has only one relief hole and six charged holes. According to Kilström, it is the most efficient cylinder cut. A modified double spiral cut was drilled: it had two relief holes instead of one and the hole spacing was a bit wider. The first double spiral cut was loaded with TRIMRITE cartridges. The second one had only one hole loaded with continuous TRIMRITE of 25 mm diameter.

The first five holes can easily be seen on the video tapes; they produce a small flame of short duration. The sixth and the seventh hole from the modified double spiral cut are not visible because the dust is too thick. The double spiral cut pulled well to the bottom. The second one with the hole loaded with continuous TRIMRITE did not break except a length around 1.5 feet at the face. The hole diameter is 37 mm and the continuous TRIMRITE is 25 mm diameter. The decoupling was apparently too high for this application.

Week 9

- October 11 blast

With the success gained with long delay and the double spiral cut, it was decided to try the corkscrew with long delays. The cut was loaded with TRIMRITE cartridges. Three holes are visible on the video recording. The cut pulled well but there were some large pieces of broken rock at the bottom. The third hole did break at the back but left around a foot of solid rock at the face. It is probable that the last cartridge of the explosive column was blown out of the hole by an adjacent hole.

- October 12 blast

A round with a double spiral cut was blasted in the 251-8 south stope. It was in ore across grain. The double spiral cut had two relief holes. The water spray was used and no sulfide dust explosion occurred. The video had a transmission problem and the record was very bad. The round pulled well and there was no bootleg present in the cut area. No stemming was used.

Week 10

- October 16 blast

A slash was blasted in 251-8 south stope. It consisted of seven columns of four holes all blasted with fast caps. It was filmed to record how the material was moving and the extent of the flame. From the recording the B-LINE ignition and a small flash from the first hole can be seen. The rest of the record is unusable because a rock hit the LEXAN window in front of the camera and broke the window's support flange. The camera lens was damaged in the process. The slash itself was successful with good fragmentation and excellent pull.

On the same shift a round on 2 sub north was monitored by the vibration monitoring crew. The blast was in a brittle dyke formation and the standard cut was used (however the four cut holes were loaded with TRIMRITE cartridges). The results were average as back control problems were experienced (this was common in this geology formation).

- October 17 blast

A round was taken in the hanging wall drift; ore, along grain. A double spiral cut with one relief hole was used. The water spray was attached to the roof. No sulfide dust explosion occurred. No bootlegs were present in the cut. There was around 10 inches of bootlegs in a few side wall holes.

After this blast some problems arised. The scissor lift broke down which forced the bolting cycle to a halt. As a result no active faces were left to drill and blast and the tests were halted for three days. Then the jumbo was taken out for repairs and the work ceased again for a further week.

- October 20 blast

A round was already drilled in the 1000 decline access. The geology was different since it was in the footwall sedimentary rock. The cut was loaded with TRIMRITE. The round broke well and the back was of a better quality than in the ore.

Week 11

- October 23 blast

A ventilation bulkhead was constructed in the hanging wall in order to complete the breakthrough round in the 249-8 south stope. The material was ore and across grain. The black and white camera was used to film the blast because the color one was

out of order. A three section cut was used. All holes were loaded with ANFO. The water spray was installed on the floor and aimed directly at the cut.

A small sulfide dust explosion occurred at 1.89 seconds after the ignition of the blast (delay #5). The flame lasted only a few tenths of a second and subsided before re-igniting at 2.61 seconds (#8 delay). The explosion lasted for a further two seconds before the video went blank (a burnt coaxial cable). Only two lengths (200 feet) of vent tubing were burnt. Fragmentation and back conditions were good. The camera was damaged beyond repair during this explosion.

- October 26th blast

The jumbo was available on the afternoon. A double spiral cut was drilled in the dyke formation of the north access drift. The six inner holes were loaded with TRIMRITE and the six outer with ANFO. The results were not good: the helpers (six outer holes) had benched around 3 feet.

- October 28 meeting

A meeting with JKMRC was held at the mine on the 28. The contract with JKMRC was completed and all loose ends had to be taken care off. The JKMRC team left the site on October 30.

Week 12

- October 31 blast

A quick setup to install the hose near the back was designed by the mining crew. It consisted of eye bolts spaced 6 to 8 feet apart with the hose slipped inside and pulled 6 feet near the face. The eye bolts are inserted during the rockbolting operation and are also used to hang the vent tubing. After a blast, all that is required is to slide the hose further and the setup is ready for the next blast.

The round was in ore across grain. It was all loaded with ANFO. The fire nozzle was on the hose. The hose was hung up 12 feet from the face and 6 feet from the right wall. The water spread around 3 feet above the cut and the spray was around 5 feet wide at the face. A sulfide dust explosion occured, burning 5 length of vent tubing. The hose was not damaged by the fly rocks. It is believed that because the fire nozzle was not aimed properly it may have been sufficient to cause the explosion.

- November 2 blast

A round was blasted in the dyke formation in the 2 sub north access. The cut holes were loaded with LOMEX, a low strength explosive that can be loaded pneumatically like ANFO. The long delays were used with boosters to make sure that the explosive detonated. The blast was not very successful as .4 meter bootlegs were left in the cut. The rock in this dyke formation is very difficult to blast and also it is hard to control the back.

- November 3 blast

A round was blasted in the 251-8 south stope. The rock was ore and across grain. The water was aiming at the back and was falling in a curtain of water 5 feet from the face. The hose was hung up to the back, 8 feet from the face. No sulphur ignitions were monitored and no bootlegs were observed.

Week 13

- November 6 blast

A round was blasted in the 253-8 south stope access. It was in the dyke formation. The cut was charged with LOMEX and ignited with boosters. Only small bootlegs were produced.

The rest of the double spiral cut (October 6 blast) with two reamer holes was loaded with LOMEX and blasted. Small boosters were used. Don Gagnon's camera and recorder were used to film the flame produced by the LOMEX. No significant differences were noticed between LOMEX and ANFO.

- November 7 blast

Four holes, fully confined and loaded with LOMEX, were fired in the 249-8 north stope south wall. Different collars were used to verify the difference in flame extent and duration. The water spray was also used in this test. A Sympathetic detonation occurred in the blast due to the proximity of the holes. The hole #4 appears to have deflagrated instead of detonating. The video results were more or less good because of the dust produced by the first hole. The water-spray definitely appeared to reduce the flame duration and extent.

- November 8 blast

A round was blasted in the hanging wall drift. The rock was ore and along grain. The cut was loaded with LOMEX. The hose was fixed to the back 6 feet from the face. It sprayed over the

cut and cascaded down on the face. A good blast performance was obtained and no sulphur ignitions were monitored. The hose was intact.

- November 9th blast

A round was blasted in the 2 sub north dyke formation. Eight back holes all loaded with LOMEX were used in order to improve the back control. This did not stabilize the back significantly however.

- Control slash

Four holes were blasted in the 249-8 north stope to verify the flame extent of LOMEX and the importance of the collar. Again the tests were not conclusive because the lamp set was smashed by the first hole and the dust was too thick to see the second hole.

This completes the tests performed under this study. The mining crew has agreed to pursue the spray tests in order to evaluate its performance over an extended period of time.